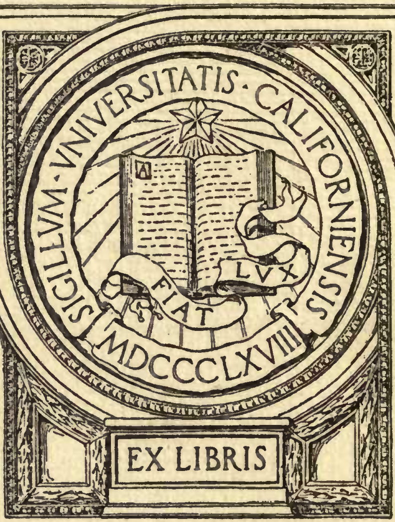
ELECTRO·METALLURGY

JOHN·B·C·KERSHAW

WESTMINSTER SERIES



SIGILLVM · VNIVERSITATIS · CALIFORNIENSIS
FIAT LVX
MDCCCLXVIII
EX LIBRIS

Mining dept.

The D. Van Nostrand Company
intend this book to be sold to the Public at the advertised price, and supply it to the Trade on terms which will not allow of reduction.

2-17 11-1M

SOLD BY
BOOK DEPARTMENT
MINING and SCIENTIFIC PRESS
667 HOWARD STREET
SAN FRANCISCO

The "Westminster" Series

MINING
U. OF C.
ASSOCIATION

ELECTRO-METALLURGY

ELECTRO-METALLURGY

BY

JOHN B. C. KERSHAW

FELLOW OF THE INSTITUTE OF CHEMISTRY, MEMBER OF THE FARADAY SOCIETY, ETC., ETC.

With 61 *Illustrations*

UNIV. OF
CALIFORNIA

NEW YORK
D. VAN NOSTRAND COMPANY
23 MURRAY AND 27 WARREN STREETS
1908

TN685
K4
MINING DEPT.

BRADBURY, AGNEW, & CO. LD., PRINTERS,
LONDON AND TONBRIDGE.

Mining dept

PREFACE

THE author's aim in this book has been to give a brief and clear account of the industrial developments of electro-metallurgy, in language that can be understood by those whose acquaintance with either chemical or electrical science may be but slight.

No attempt therefore has been made to discuss the theoretical side of the subject. The meanings of the few technical terms and expressions that have almost necessarily crept into the book will be found in a glossary on p. 299, where they can easily be referred to by readers.

The facts and figures recorded in the book relate to the position of the electro-metallurgical industries at the end of 1906, unless otherwise stated. The developments in some of these industries are, however, extremely rapid, and allowance must be made for the difference in date when comparing the figures given with those published more recently.

The patent literature of electro-metallurgy is unusually extensive and voluminous, and no useful purpose would have been served by digging deep into it. The author has therefore confined himself to descriptions of apparatus and processes which have once received practical trial upon an industrial scale, or which are now in commercial use.

For aid, either in the form of facts and figures, or of photographs and prints, the author's thanks are due to the following:—

Mr. James Swinburne, M. Paul Heroult, Dr. H. Goldschmidt, Messrs. Keller, Leleux & Cie., The British Aluminium Co., the Editor and Publishers of "Electro-chemical and Electro-metallurgical Industry," and the Editor and Publishers of the "Scientific American."

JOHN B. C. KERSHAW.

THE WEST LANCASHIRE LABORATORY,
WATERLOO, LIVERPOOL,
January, 1908.

CONTENTS

CHAPTER I.

PAGE

INTRODUCTION AND HISTORICAL SURVEY 1

CHAPTER II.

ALUMINIUM.

I. INTRODUCTION 10

II. PRODUCTION AND PRICE 11

III. DETAILS OF PROCESSES AND WORKS 14

The Cowles Process—The Hall Process—The Heroult Process — Foyers, N.B. — La Praz, France — Massena, U.S.A.—Shawinigan Falls.

IV. COSTS 32

V. UTILISATION 33

Electrical Conductors — Foundry and Metallurgical Uses—Printing—Alloys—Miscellaneous Uses.

VI. FUTURE OF THE METAL 4

CHAPTER III.

Bullion and Gold.

PAGE
I. INTRODUCTION 46
II. HISTORICAL NOTES 47
III. SILVER REFINING PROCESSES 50
The Moebius Process — The Balbach Process — The Dietzel Process—The Betts Process.
IV. GOLD REFINING PROCESS 59
The Wohlwill Process.
V. GOLD EXTRACTION PROCESSES 62
The Siemens & Halske Process — The Greenawalt Process.

CHAPTER IV.

Calcium Carbide and Acetylene Gas.

I. HISTORICAL INTRODUCTION 65
II. THE CARBIDE FURNACE AND PROCESS 69
III. PRODUCTION 74
IV. UTILISATION 80
The Acetylene Industry — The Calcium Cyanamide Industry.

CHAPTER V.

Carborundum.

I. HISTORICAL INTRODUCTION 90
II. DETAILS OF MANUFACTURE 94
III. PROPERTIES AND USES 97

CHAPTER VI.

COPPER.

PAGE

I. INTRODUCTION 100

II. COPPER REFINING 101

Historical Notes—The Refining Process.

III. DESCRIPTIONS OF REFINERIES 109

Pembrey, South Wales — Froghall, Staffordshire — Widnes, Lancashire—Swansea—Newark, U.S.A.—Great Falls, U.S.A.—Anaconda, U.S.A.—Perth Amboy, U.S.A. —Hamburg, Germany—Eisleben, Germany.

IV. COSTS 121

V. PROPERTIES AND UTILISATION 122

VI. THE ELMORE AND SIMILAR PROCESSES 124

Leeds—Schladern—Dives—The Dumoulin Process—Miscellaneous Processes.

VII. ELECTROLYTIC EXTRACTION PROCESSES 134

The Hoepfner Process—The Carmichael Process.

VIII. ELECTRO-METALLURGICAL CONCENTRATION PROCESS . . . 137

CHAPTER VII.

FERRO-ALLOYS.

I. INTRODUCTION 139

II. DESCRIPTIONS OF WORKS 143

Kerrousse and Livet—Ugine, Courtepin and Montbovon —La Praz—Kanawha Falls.

III. UTILISATION 154

CHAPTER VIII.

Glass and Quartz Glass.

PAGE
I. INTRODUCTION 157
II. GLASS 158
III. QUARTZ GLASS 163

CHAPTER IX.

Graphite.

I. INTRODUCTION 167
II. DETAILS OF PROCESS 170
III. UTILISATION 172

CHAPTER X.

Iron and Steel.

I. INTRODUCTION 175
II. DESCRIPTIONS OF FURNACES AND PROCESSES . . . 178
Stassano—Heroult—Keller—Kjellin—Gin—Girod.
III. YIELDS AND COSTS 202
IV. COMPARATIVE COSTS 206

CHAPTER XI.

LEAD.

I. INTRODUCTION 209
II. THE SALOM PROCESS 210
III. THE BETTS REFINING PROCESS 211
IV. THE BETTS REDUCTION PROCESS 214
V. WHITE LEAD PROCESSES 215

CHAPTER XII.

MISCELLANEOUS PRODUCTS.

PAGE
I. INTRODUCTION 216
II. CALCIUM 217
III. CARBON BISULPHIDE 219
IV. CARBON TETRA-CHLORIDE 221
V. DIAMANTINE 223
VI. MAGNESIUM 223
VII. PHOSPHORUS 224
VIII. SILICON AND ITS COMPOUNDS 226
Metallic Silicon—Siloxicon—Silicon Copper.

CHAPTER XIII.

NICKEL.

I. INTRODUCTION 231
II. WET PROCESSES 233
The Thum Process—The Orford Process—The Browne Process—The Hybinette Process—The Hoepfner Process.
III. DRY PROCESSES 241
The Clergue-Horry Process—The Heroult Process.

CHAPTER XIV.

SODIUM.

I. INTRODUCTION 244
II. DESCRIPTIONS OF CELLS AND PROCESSES 248
Castner — Rathenau & Suter — Boelsterli — Becker — Darling—Ashcroft.

CHAPTER XV.

TIN.

PAGE
I. INTRODUCTION 256
II. ALKALINE PROCESSES FOR TIN STRIPPING 259
Beatson—Goldschmidt.
III. ACID PROCESSES FOR TIN STRIPPING 261
Gelstharpe—Smith & Gutensohn.
IV. SALT PROCESSES FOR TIN STRIPPING 263
Claus—Garcia—Brown & Niel—Bergsoe.

CHAPTER XVI.

ZINC.

I. INTRODUCTION 269
II. WET PROCESSES 271
The Hoepfner Process—The Ashcroft Process—The Siemens & Halske Process—The Cowper-Coles Process.
III. DRY PROCESSES 277
The Swinburne-Ashcroft Process—The Steinhart & Vogel Process.
IV. ELECTRO-THERMAL PROCESSES 286
The Casoretti & Bertani Furnace—The De Laval Furnace—The Salgué Furnace—The Brown & Oesterle Process.
V. ELECTRO-GALVANISING 293
Cowper-Coles Process—Alexander Process—Richter Process.

GLOSSARY 299

LIST OF ILLUSTRATIONS

FIG. PAGE

1. COWLES ALUMINIUM FURNACE 14
2. HEROULT EARLY ALUMINIUM FURNACE 18
3. HEROULT LATER ALUMINIUM FURNACE, DIAGRAM OF . . 20
4. LOCH NESS AND ALUMINIUM WORKS: DIAGRAM . . . 21
5. INTERIOR OF POWER HOUSE, FOYERS 23
6. LA PRAZ ALUMINIUM WORKS 25
7. PIPE LINE AT LA PRAZ 26
8. PLAN OF POWER DEVELOPMENT, MASSENA 28
9. CANAL CUTTING AT MASSENA 29
10. USE OF "THERMIT" FOR REPAIRING STEAMER . . . 36
11. USE OF "THERMIT" FOR REPAIRING FLY-WHEEL . . 38
12. WATT'S METAL REFINING VAT 48
13. MOEBIUS SILVER REFINING VAT 52
14. BALBACH SILVER REFINING VAT 55
15. CONTINUOUS CARBIDE FURNACE (MEMMO TYPE) . . . 70
16. INTERMITTENT CARBIDE FURNACE (MEMMO TYPE) . . . 72
17. INTERMITTENT CARBIDE FURNACE (CARLSEN TYPE) - . 74
18. THE VOLTA CARBIDE WORKS AT VERNIER, NEAR GENEVA . 79
19. ACETYLENE GENERATOR (SIMPLEST TYPE) 81
20. CENTRAL ACETYLENE GENERATING PLANT AT OLIVA, GERMANY 82

FIG. PAGE
21. THE LEWIS ACETYLENE LAMP 84
22. THE ROSS ACETYLENE LAMP GENERATOR 85
23. ACHESON CARBORUNDUM FURNACE 93
24. THE MULTIPLE SYSTEM OF ELECTRODE ARRANGEMENT . . 104
25. THE SERIES SYSTEM OF ELECTRODE ARRANGEMENT . . 106
26. FORM OF ANODE SLAB 107
27. THE TANK HOUSE AT THE WIDNES REFINERY . . . 112
28. WALKER & THUM ANODE CASTING MACHINE 114
29. THE TANK HOUSE AT RARITAN COPPER REFINERY, U.S.A. . 117
30. THE TANK HOUSE AT LEEDS REFINERY 125
31. CASTING MACHINERY FOR REFINED COPPER PIGS . . . 137
32. KELLER FURNACE FOR FERRO-ALLOYS 144
33. THE KERROUSSE WORKS, GENERAL VIEW 146
34. GIROD FURNACE, VERTICAL SECTION AND PLAN . . . 149
35. GIROD FURNACE, TILTING TYPE 150
36. ELECTRIC GLASS FURNACE (FRONT VIEW) 159
37. ELECTRIC GLASS FURNACE (BACK VIEW) 160
38. QUARTZ LABORATORY VESSELS 163
39. ACHESON GRAPHITE FURNACE 168
40. SIEMENS CRUCIBLE FURNACE 176
41. STASSANO ROTARY FURNACE 180
42. STASSANO FIXED TYPE OF FURNACE 182
43. HEROULT CRUCIBLE FURNACE 184
44. HEROULT CRUCIBLE FURNACE (DISCHARGING) . . . 186
45. KJELLIN FURNACE 195
46. GIN FURNACE 198
47. TAYLOR CARBON-DISULPHIDE FURNACE 219
48. MACHALSKE CARBON-TETRACHLORIDE FURNACE . . . 222

FIG. PAGE
49. HYBINETTE VAT FOR NICKEL DEPOSITION 238
50. VAT ROOM AT PAPENBURG NICKEL WORKS 240
51. CASTNER SODIUM CELL . . . - 248
52. BITTERFELD ELECTRO-CHEMICAL WORKS (GENERAL VIEW) . 250
53. BITTERFELD ELECTRO-CHEMICAL WORKS (ENGINE ROOM) . 251
54. ASHCROFT SODIUM CELL 254
55. GOLDSCHMIDT TIN-STRIPPING WORKS (GENERAL VIEW) . . 259
56. GOLDSCHMIDT TIN STRIPPING WORKS (ENGINE ROOM) . . 260
57. SWINBURNE-ASHCROFT TRANSFORMER FOR ZINC EXTRACTION . 279
58. SWINBURNE-ASHCROFT TRANSFORMER 280
59. CASORETTI & BERTANI ZINC EXTRACTION FURNACE . . 287
60. DE LAVAL ZINC EXTRACTION FURNACE 288
61. COWPER-COLES ELECTRO-GALVANISING PLANT . . . 295

ELECTRO-METALLURGY.

CHAPTER I.

INTRODUCTION AND HISTORICAL SURVEY.

Electro-Metallurgy may be defined as that branch of metallurgy which deals with the extraction and refining of metals by aid of electricity. The science and theory of the subject date back to the early years of the nineteenth century. The practical applications and developments of electro-metallurgy are comprised almost entirely within the last twenty-five years. In that time remarkable progress has been made, and indications are not wanting to show that as the coalfields of Europe and America become gradually exhausted, electricity generated from water-power will take a more and more prominent place in the metallurgy of the future. This will lead to a transfer of great industries and populations, of enormous importance for the countries and nations concerned. Already electricity is an indispensable agent in the aluminium, bullion, copper, ferro-alloys and sodium industries; while calcium carbide, graphite, quartz-glass and other similar products can be manufactured only by the aid of electric heat. The latest developments are to be found in the iron and steel industry, where electric furnace methods are now making considerable headway. The electric refining furnace is, in fact, likely to become a common and useful adjunct of the blast-furnace of the future. The waste gases of the latter will then be utilised to generate the electric energy required for the refining process, and for the production of special alloys of iron and steel with the rare metals.

Electric energy may be utilised in the domain of applied metallurgy for three purposes. *First:* It may be employed as the agent by which chemical combinations of the metals with other elements are broken down and separated into simpler fragments, generally represented by the metal and some gaseous element. This application is named *electrolysis.* It may be carried out either with aqueous solutions of the metallic salt, as in the processes for obtaining pure *nickel;* or with fused mixtures, as in the manufacture of *aluminium* and *sodium.* *Second:* Electric energy may be employed for attacking an impure metal and for carrying off the one constituent, while leaving the other metals, present as impurities, unattacked. The actual process of separation used here is again one of electrolysis; but the electric energy first aids in obtaining the metal in combination with the solvent, and then splits up this combination into its constituent parts. The methods by which bullion and copper are refined, and the tin-stripping industry, are examples of this application.

The third method by which electric energy is utilised in metallurgical operations is simply as a thermal agent; the electric current being employed to generate heat. The thermal effect may be either external or internal; that is to say, it may be obtained either by radiation from the electric arc or from some resistance material such as *kryptol*, or by utilising the electrical resistance of the material itself. The calcium carbide, carborundum, ferro-alloys, quartz-glass and steel industries illustrate this use of electricity.

Turning now to a consideration of the historical development of electro-metallurgical science and industry in the United Kingdom, it is to be noted that many of the more important observations and discoveries relating to the action of the electric current upon chemical elements and compounds were made by Davy, Nicholson, Carlisle and Cruikshank, between the years 1800 and 1808.

In the year 1800, Nicholson and Carlisle decomposed water by the aid of an electric current generated from a voltaic pile, and showed that an acid was formed at the positive pole of their cell. In the same year, Cruikshank proved that metals could be deposited from solutions of their salts, by use of the same agency, and he suggested that this method should be employed for analytical work.

Davy, who commenced to experiment with the electric current in 1806, was, however, the most successful of these early workers, and in the years 1806—1808, he carried out a series of striking experiments and discoveries. In 1806 he settled the vexed question as to the cause of the acid formation at the anode, observed by Nicholson and Carlisle, and also showed that many salts could be split up into their component parts of acid and base, by aid of the electric current.

In 1807, he decomposed the alkali hydrates by the same agency, and obtained a world-wide reputation and renown by the separation and discovery of the alkali metals—potassium and sodium. It is not generally known that it was only by persistent work and by scientific reasoning, that Davy achieved his object in this research. Two attempts to separate these metals by electrolysis had failed before Davy perceived and applied the right method of solving the problem, and a chemist with less ability or scientific insight might have missed altogether the discovery, which raised Davy to a foremost place among the scientific workers of the last century.

Following up his discovery of potassium and sodium, Davy, in 1808, produced the carbide of the former metal by aid of the electric arc, and also separated calcium, barium, strontium, and magnesium from the alkaline earths, by use of the method first applied by Berzelius, in which mercury was employed as negative electrode, and formed an amalgam with the metal separated.

Davy was followed, in this new branch of study, by his former pupil and assistant, Faraday. The results of the experimental work of the latter, in the laboratories of the Royal Institution, London, in the period of 1812—1840, placed the science of electro-metallurgy upon a firm basis of accurately observed phenomena, and led to the enunciation of laws, which more than half a century of progress has still left intact.

Although modern theories of electricity and electrolytic action are not those formulated by Faraday, the change in theory has not vitiated any of the early results, and *Faraday's Law* is still one of the fundamental principles of electro-metallurgical science.

The middle years of the nineteenth century were not marked, in the United Kingdom, by any very striking discoveries in this domain. Gore, about 1850, separated aluminium from its chloride by electrolysis of the fused salt, and obtained silicon from potassium silicate by the same means. In 1858, the same worker also separated and discovered *explosive antimony*.

As regards industrial developments, Jacobi, Murray, and Spencer were the first to point out that the electric current could be employed for copying moulds and casts in copper, and in 1839, these three workers—the first two in London and the third in Liverpool—laid the foundation of the modern industry of electro-typing.

In 1840, Woolrich, of Birmingham, started in that city a small works, for making magneto-electric machines or dynamos, and showed how the current from these could be employed for the electro-deposition of silver. This was, in fact, the commencement of the large and important electro-plating industry, which now flourishes in the midland city. Woolrich's original electro-magnetic machine has been bought and is now preserved by the Birmingham

City Corporation, as an interesting and valuable relic of the past.

In 1851, Charles Watt took out the remarkable Patent No. 13,755 which may rightly be considered as the master patent of the electro-chemical and electro-metallurgical industries in the United Kingdom.

In this patent, Watt described in some detail how the electric current might be employed for producing alkali hydrates and chlorine, hypochlorites, or chlorates from solutions of the alkali metal chlorides, and how it might be utilised for refining copper, silver or other metals, or for separating these from their ores. This patent is well worth study by all interested in the historical development of electro-chemical and electro-metallurgical industry in the United Kingdom.

Watt's ideas, as put forth in this patent of 1851, could not, however, receive practical trial until the dynamo was developed and improved, and it was not until 1869, when Elkington erected the first electrolytic copper refinery at Pembrey, in South Wales, that the industrial development of the facts and ideas, gathered during the previous years of the century, commenced.

In 1870—1872, Gramme introduced improvements into the design and construction of the armature rings for dynamos, which greatly improved their efficiency, and since that year the development in both engine and dynamo building has kept pace with the requirements of the electro-chemical and electro-metallurgical industries.

The copper refining industry was the first of these to advance. Other refineries were started in England after the early trials at Pembrey proved that the electrolytic method could be employed for this work with success, and the industry spread from England to France, Germany, and the United States.

The manufacture of aluminium was commenced in England in 1886, the Cowles process being started at Milton, in Staffordshire, in that year. The Hall electrolytic process was introduced in 1890, and the Heroult process in 1896.

The calcium carbide, nickel, tin, and zinc industries have been developed more recently, and show less development than those already named. The total number of electro-metallurgical undertakings at present in operation in the United Kingdom is between twenty and thirty, and more than two-thirds of these are working upon a fairly large scale of operations. This total does not include the very large number of installations for electro-typing and electro-plating work, which are quite distinct branches of industry.

These historical introductory notes prove that the United Kingdom has occupied a not unworthy position in the development of the electro-metallurgical industries. As regards the early discoveries and pioneer work in these branches of science, English chemists and investigators have played a very important and in fact brilliant part, the names of Davy and of Faraday being indissolubly connected with the early discoveries and historical development of the new science.

The practical developments of the electro-metallurgical industry in the United Kingdom are also of greater importance, and are more numerous than is generally realised; and taking into account the disabilities under which English inventors and manufacturers have worked in developing these new industries, the practical results are certainly striking.

The United Kingdom, like Germany, is handicapped by the fact that the country possesses few water powers which could be developed for power purposes at moderate cost.

With two exceptions all the electro-chemical and electro-metallurgical manufactures now being carried on in the United Kingdom are operated by steam power, and the products of these have to compete in the open market with the products of the water-power centres abroad. The two exceptions are the Aluminium Works at the Falls of Foyers in Scotland, and the Carbide Works at Askeaton in Ireland, which are both operated by water power. Had England possessed, like America, a vast source of power equal to Niagara, or like France, Switzerland, or Norway, a very large number of smaller waterfalls, all suitable for power development at a low capital cost, there is little doubt that her actual achievements in these new branches of industry would have been still more creditable. The fact that many of the processes that are now being worked with most successful results in other countries are based upon discoveries made by English scientists and inventors, and that in some cases these processes were first worked upon an industrial scale in the United Kingdom, proves that with cheap power and adequate supplies of raw material, much greater developments would have occurred within the British Isles.

Whether the attempts which are now being made in many quarters, to utilise the wasted energy of blast-furnace gases for electric power generation on a large scale, will redress the balance in favour of the two older manufacturing countries, remains for the future to disclose. In England and Germany, at present, many hundreds of thousands of horse power are annually wasted with these gases. In a few cases they are already being utilised in large gas engines of special design, for the mechanical work of operating the blowing engines which provide the blast.

It is but a step from this application to the utilisation of these gases for driving dynamos, and should the capital outlay upon such large gas-engine plants, and the cost of upkeep not prove excessive, it would seem highly probable that within the next ten years, great power developments of this kind will occur in both England and Germany. The disability under which each country is at present labouring with regard to the development of these new industries will then disappear, and in both countries a further expansion of those electro-metallurgical industries and manufactures, for which cheap power is an essential of success, is likely to occur.

CHAPTER II.

ALUMINIUM.

I. INTRODUCTION.

THE manufacture of aluminium upon an industrial scale by aid of the electric current dates from 1886, in which year the brothers Cowles commenced to manufacture ferro-aluminium and other aluminium alloys at Lockport in America, and at Milton in Staffordshire, England. These two works were operated until 1892. Pure aluminium could not be produced, however, by the Cowles process, and, owing to this fact, and to the rapid fall in price of the new metal, as a result of the successful operation of the Hall process in America and of the Heroult process in Europe, the Lockport and Milton works were shut down in the latter year.

Several other processes for the production of aluminium by electrolytic or electro-thermal methods have been tried experimentally since the Cowles works were started in 1886, but only two of these have survived the competition which has resulted. At the present time, the whole of the world's output of aluminium is produced either by the Hall or by the Heroult process.

The most striking failure to maintain its position occurred in connection with the development of the Webster and

Castner process at Oldbury, Birmingham, by a company named the Aluminium Company, floated in 1887. This process was not dependent upon the use of the electric current, since the reduction of the aluminium compounds was effected by the action of metallic sodium upon a mixture of aluminium fluoride and chloride at a red heat.

A large works was erected by the company named above at Oldbury, near Birmingham, and in 1888, when this works started manufacturing operations, a prosperous future was anticipated for the process and company which operated it. Three years later, or in 1891, the Oldbury works had to suspend the manufacture of aluminium by the Castner and Webster process, for the Heroult process in Europe and the Hall process in America had been developed upon an industrial scale; and the figures given on the next page show how rapidly the price of aluminium had fallen as a result of the competition of the two new processes.

II. Production and Price.

The manufacture of aluminium by the Hall and Heroult methods was commenced at New Kensington in America, in the year 1888, and at Neuhausen in Switzerland, in the year 1889. The processes were worked out independently by Hall in America and by Heroult in France, but, as now operated, they are practically identical, and consist in the electrolysis with carbon electrodes of aluminium oxide held in solution in a fused bath of cryolite and fluor-spar.

Since the introduction of the electrolytic method of manufacture, the production of aluminium has increased from 85 tons in 1889 to 12,000 tons in 1906.

The following tabular statement shows the gradual increase in output and fall in price, which has marked the industrial development of the electrolytic process:—

Year.	1885.	1886.	1887.	1888.	1889.	1890.	1891.	1892.
Production	3·12	14·8	22·5	39·4	85·2	174·5	324·5	485
Price	600	500	400	250	125	100	25	32

Year.	1893.	1894.	1895.	1896.	1897.	1898.	1899.	1900.
Production	713	1,057	1,129	1,755	3,327	3,953	5,459	7,192
Price	—	25	22	17	17·5	16·2	16·2	16·0

Year.	1901.	1902.	1903.	1904.	1905.	1906.
Production	7,420	7,750	8,102	8,550	9,000	12,000
Price	15·5	15·5	15·5	15·5	16·0	21·40

NOTE.—The production is given in tons of 2,240 lbs., and the price in pence per lb. From 1902 onwards the production figures are estimated, and from 1897 to 1904 the figures for price are based on the American values.

The manufacture of aluminium is now carried on in thirteen works—controlling over 84,000 h.p.; details of these, so far as they are known, are given on p. 13.

Assuming that 4 h.p. are required for one year to produce one ton of aluminium, the aggregate power available in these works would suffice to produce 21,000 tons of the metal per annum. Owing, however, to the diminished power available during the summer drought and to other causes, the maximum total of power is not available for the manufacture all the year round, and the 1906 production is estimated by the writer to have been about 12,000 tons.

Name of Company.	Locality of Works.	Power.
		h.p.
1. British Aluminium Company.	1. Foyers, N.B.	5,000
	2. Stangfjord, Norway	10,000 (?)
	Switzerland, in course of erection.	(?)
2. Société Electro-métallurgique francaise.	3. La Praz	7,500
	4. Gardannes	7,500
3. Compagnie des Produits chimiques d'Alais.	5. Calypso	10,000
	6. St. Felix	2,500
4. Aluminium Industrie Aktiengesellschaft.	7. Neuhausen	5,000
	8. Rheinfelden	5,000
	9. Lend Gastein	15,000
5. Pittsburg Reduction Company.	10, 11. Niagara Falls	10,000
	12. Shawinigan Falls (Canada).	5,000
	13. Massena (U.S.A.)	12,000

When one recalls the fact that down to 1886 aluminium was merely a laboratory curiosity, and was sold at 3*s.* 6*d.* per ounce, or 56*s.* per lb., the progress made during the last twenty years in cheapening the metal and rendering it available for use in the arts and industries is certainly remarkable.

III. DETAILS OF PROCESSES AND WORKS.

The Cowles Process.—The Cowles process of aluminium production, which was worked upon a fairly large scale in England and America during the early days of the industry, depended upon the reducing action of carbon upon alumina (Al_2O_3) at a red heat. The process was electro-thermal and not electrolytic in character, the electric current being employed merely as a heating agent. The reduction of alumina by carbon alone is not possible in the absence of any metal with which the aluminium when reduced can alloy itself, for aluminium at all temperatures has a greater affinity than carbon for oxygen. But in the presence of copper or iron, the reduction of alumina by carbon is practicable in the electric furnace, since the affinity of the two metals and disposition to form an alloy renders the aluminium more ready to part with its oxygen.

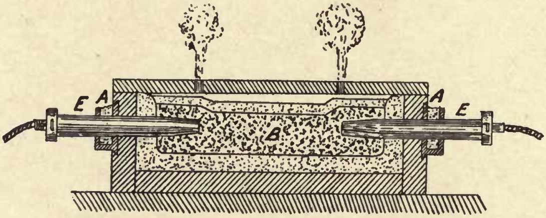


FIG. 1.—Cowles aluminium furnace.

The Cowles process as worked at Milton, in Staffordshire, consisted in heating powdered corundum (a form of alumina, Al_2O_3) with charcoal and iron filings in a large electric furnace (Fig. 1). Charcoal coated with lime was used as lining for the furnace. A current of 5,000 amperes at 60 volts was employed for heating the furnace, and one and a half hours was required for reducing each charge. The yield was stated to be 1 lb of alloy per 18 e.h.p. hours.

The dynamo used at Milton for supplying the electric current to the Cowles furnace was of 300 k.w. capacity, and at the date of this trial (1888) it was the largest generator yet built in any country. It was visited and inspected by many engineers as a notable triumph in electrical engineering work. As 5,000 and 7,500 k.w. generators are now common, the Milton dynamo serves only to mark an epoch in dynamo construction.

A similar furnace was in operation at Lockport, in U.S.A., from 1886 to 1892, but owing to the rapid development of the Hall and Heroult processes and to the fact that pure aluminium could not be produced by the Cowles furnace, both plants ceased work in the latter year.

The Hall Process.—The Hall process of aluminium reduction has been operated in America since 1888. It is at present employed at three works in that country, disposing of 24,000 h.p., while a fourth works is now in course of construction. The Hall process is also employed at the aluminium works situated at Calypso, near St. Michel, and St. Felix, in France. All of these aluminium works are operated by water power.

The Hall process is electrolytic in character, and depends upon the use of a fused electrolyte containing cryolite, fluor-spar, and alumina. Cryolite is a mineral containing aluminium fluoride and sodium fluoride, while fluor-spar is a crystalline variety of calcium fluoride. When these three compounds are melted together, the alumina dissolves in the mixture of fluorides, and if an electric current be now passed through the mixture, using carbon as electrode material, aluminium is separated in the molten state at the negative pole. As originally carried out, Hall used a carbon-lined crucible for containing the electrolyte, and heated this externally. It was, however, found more convenient to utilise the current for melting and keeping the electrolyte at the desired temperature, and external heating of the crucibles or pots was discarded very early in the development of the process. The alumina is alone decomposed by the current if the voltage or pressure be kept below 5 volts, the fluorides of aluminium, calcium, and sodium remaining unaltered in the bath, and serving merely as a solvent or carrier for the oxide of the metal. Aluminium in the molten state is heavier than the molten mixture forming the electrolyte, and falls to the bottom of the decomposing bath, a fact that is of considerable importance to the successful conduct of the process. The oxygen liberated at the anode combines with the carbon and forms carbonic acid gas (CO_2), and since a very pure form of carbon is essential for the production of pure aluminium, the cost of anodes forms an important item in the total cost of production. Professor Richards has given the following details of the operation of the Hall process at Niagara Falls: The baths are of sheet-iron lined with carbon, and are oblong in shape. The current passes into each bath or "pot" by carbon electrodes, each 3 inches in diameter and 18 inches in length, suspended from a copper carrier bar running longitudinally over the bath. The "pots" are worked in series, and the drop of e.m.f. at each "pot" is 5 volts. Each "pot" absorbs about 65 h.p. and produces about 112 lbs. aluminium per twenty-four hours. The electrolyte is kept at the necessary degree of fluidity (850°—900° C.) by the action of the current, and about one-half of the heat energy of the electricity is required for this purpose, the other half performing the actual electrolytic dissociation of alumina into the metal and oxygen. Alumina is added to the "pots" at regular intervals as that dissolved in the electrolyte is decomposed, and the surface of the electrolyte is kept covered with powdered coke to prevent loss of heat by radiation.

The annual production of aluminium by the three works operating the Hall process in America is now about 4,200 tons, and the manufacture is still growing. It is interesting to note that Mr. Hall, the inventor, was still a student at college when struck with the possibilities of the process, and that the leading details of it were worked out and a patent applied for before Mr. Hall had completed his twenty-third year.

The Heroult Process.—The Heroult process of producing aluminium from alumina is employed at seven out of the nine European works, and is so similar in all respects to the Hall process that one is surprised to find distinct patents and organisations in existence for the two processes.

The process was worked out by M. Heroult, a French metallurgical chemist, in the years 1886–88, and manufacture upon an industrial scale was commenced at Neuhausen, in Switzerland, in the year 1889.

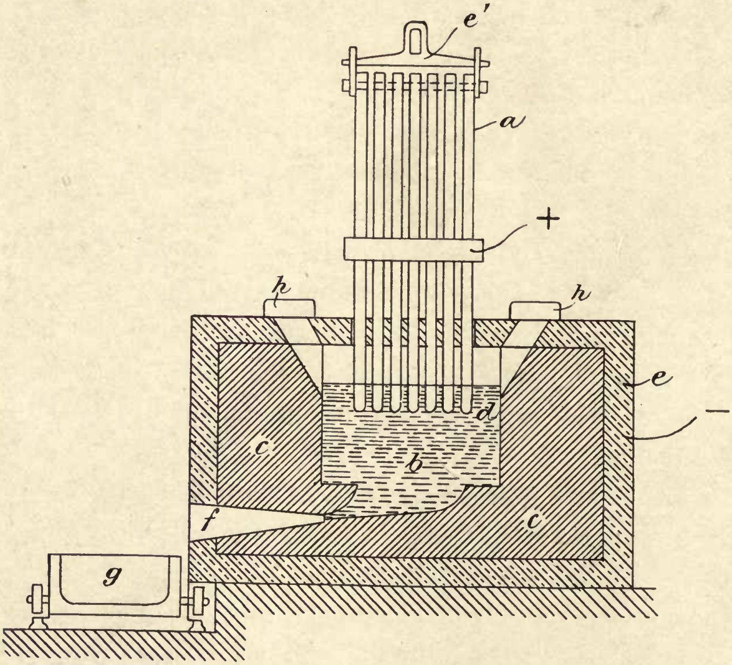


FIG. 2.—Heroult early furnace.

Until 1891 only copper aluminium alloys were produced by the Heroult process at this works, but improvements made in it by Kiliani rendered the process suitable for the manufacture of pure aluminium, and since 1891 this has been the chief product at Neuhausen and at the six other European works. In the original patent specification an iron vessel thickly lined with carbon blocks was described; the walls and base being used as the negative electrode exactly as in the Hall process.

A bundle of carbon rods or plates were suspended over this vessel, and acted as the positive electrode of the electrical circuit, the electrolyte was composed of fused alumina without any other admixture (Fig. 2). Molten copper was, however, used on the floor of the bath, and the aluminium, separated from the Al_2O_3 by the action of the current, at once united with this and formed aluminium bronze. The electrolyte was kept in the molten state by the action of the current, and external heating does not appear to have been used at any stage in the development of the Heroult process. The modifications introduced into the Heroult process since 1889 have related to the composition of the electrolyte, and as now carried out at Neuhausen, La Praz, Foyers, and elsewhere, it is practically identical in principle and operation with the Hall process. It is interesting to note that Hall's first American patent for aluminium production is dated 1886, while Heroult's first French patent for the same manufacture is dated 1887. Both inventors were at that date unaware of the other's experimental work, and they appear to have arrived independently at the same solution of the problem which they had attacked.

The English company which operates this process commenced the manufacture of aluminium at Foyers in 1896, and a great outcry was made at that time, concerning what was then called the desecration of one of the most beautiful falls in North Britain. Very large hydraulic engineering works were carried out, and 14,000 h.p. were rendered available for manufacturing operations. The original installation of electrical generating machinery was equal to 3,500 h.p., but extensions are now being carried out at Foyers which will bring this total up to from 10,000 to 15,000 h.p.

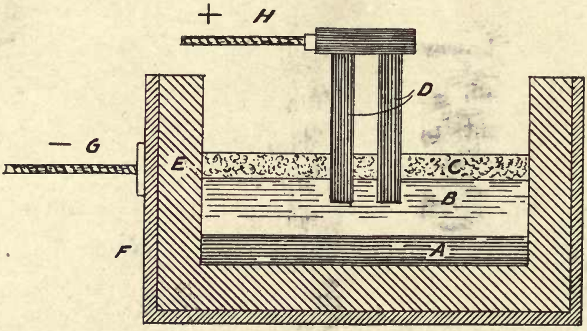


FIG. 3.—Diagram of Heroult furnace.

The other works producing aluminium by the Heroult process in Europe are situated at Rheinfelden, Lend Gastein, La Praz, Gardannes, Foyers and Sarpsfos in Norway. The Heroult works have between 45,000 and 50,000 h.p. available for the manufacture, as compared with 40,000 h.p. in the works using the Hall process.

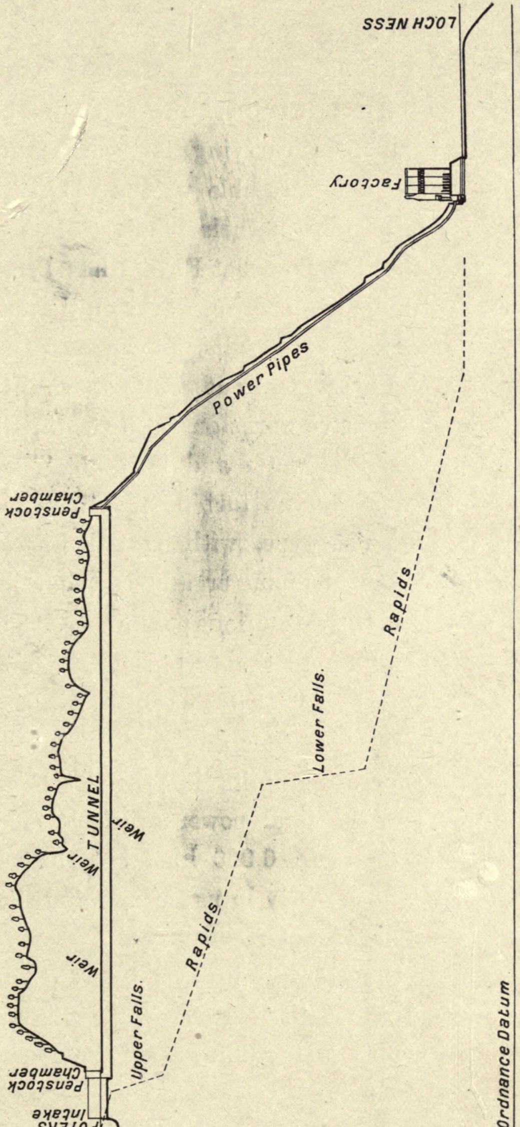


FIG. 4.—Diagram of Loch Ness and Aluminium Works.

The electrical generating plant at four of these works will be described in some detail. The actual arrangement of the decomposing pots or baths, however, cannot be described, since the companies carrying on the manufacture of aluminium are not agreeable to the publication of any details of this portion of their plant.

Foyers, N.B.—The power at Foyers has been developed by large hydraulic engineering works on the River Foyers near to Loch Ness. Two falls, known as the Upper and Lower Falls of Foyers, occur at points above this Loch, and by means of extensive works on Loch Garth, the company have impounded this water and have obtained a head of 350 feet for working the turbine plant. These turbines are of the impulse type, with partial injection. Each turbine is 10 feet in diameter and can develop 700 h.p. when running at 140 revolutions per minute. The dynamos are keyed directly on to the turbine shafts, and therefore run at the same speed. They are 24-pole machines, with armatures weighing 14 tons, and yield direct current. The number of generating units installed at Foyers is now (1906) nine, and the total power available for aluminium production is therefore 6,300 h.p., but the whole of the generating plant is rarely in use at one and the same time owing to shortage of water.

As regards the Heroult process, in its original form as already stated, the bath was charged with pure alumina, and molten copper was employed on the floor of the bath as the negative pole or cathode. In the modified process

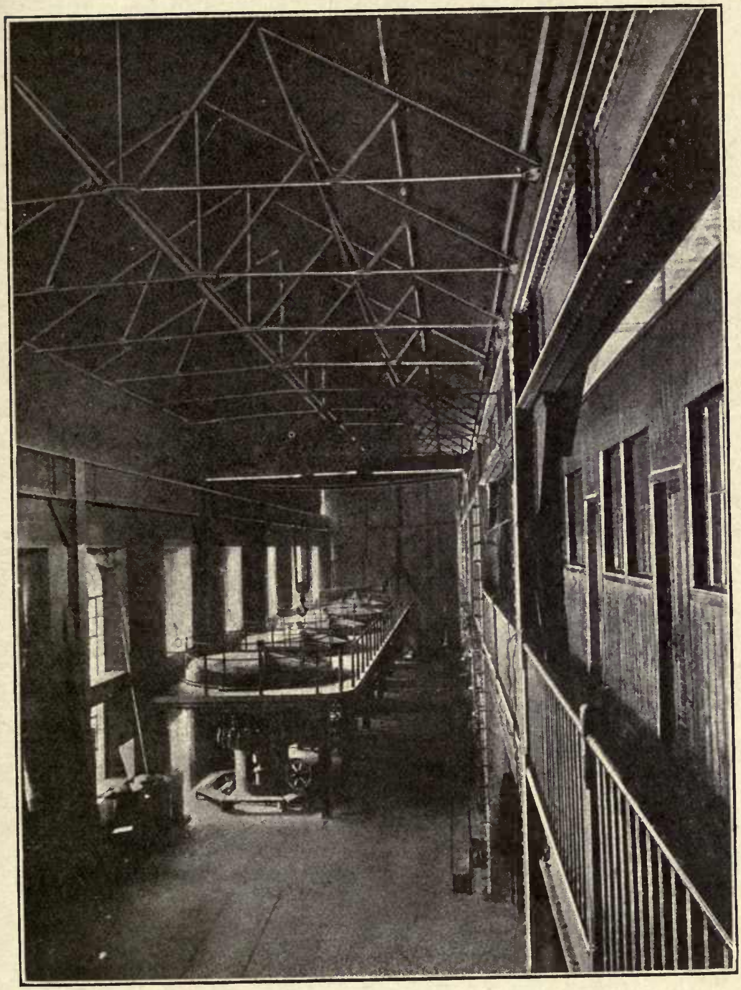


FIG. 5.—Interior of Power house, Foyers.

introduced in 1891, the copper is dispensed with, and pure aluminium is obtained on the floor of the bath in the molten state. The modification of the original process lay in the use of cryolite and fluor-spar as solvents for the alumina charged into the bath. By the use of these fluxes the temperature required to keep the bath in the molten state was greatly reduced, and the aluminium was separated as metal without any danger of its volatilisation or oxidation, and without use of iron or copper at the cathode. The Heroult process thus became in all essentials exactly similar to the Hall process; and an understanding as to the countries in which each should be operated, was arrived at by the respective companies owning the patent rights.

The refined alumina used in the Foyers works was at first obtained from Irish bauxite, prepared by the usual chemical process at the company's own factory at Larne, in Ireland. It has been stated, however, that the Irish bauxite is not adapted for producing a high grade product, and that the refined alumina used at Foyers is now obtained partly from other sources. The British Aluminium Company also own a works at Greenock for manufacture of the carbons employed in the baths, and rolling mills at Milton, in Staffordshire, where the bar alumina produced at Foyers is sent for conversion into sheets and rods. As regards yield, in the early days of the industry, 1 lb. of alumina was obtained at Foyers with an expenditure of 12 e.h.p. hours, this being equivalent to slightly over 3 e.h.p. years per ton of metal produced. The refined aluminium was reported to test as follows: 99·58 per cent. Al, ·18 per cent. Si, ·24 per cent. Fe.

La Praz, France.—This is the chief works of the *Société Electro-métallurgique française*, the company formed to operate the Heroult process in France. The works are located in the valley of the Arc in Savoy, 3,100 feet above sea-level and close to the Italian frontier. The power is obtained from the water of the Arc, by means of two dams, half a mile and 1½ miles from the power house, and by the usual conduit and pipe lines. The heads of water obtained at the power house from these hydraulic engineering works are 105 feet and 238 feet respectively. The second pipe line,

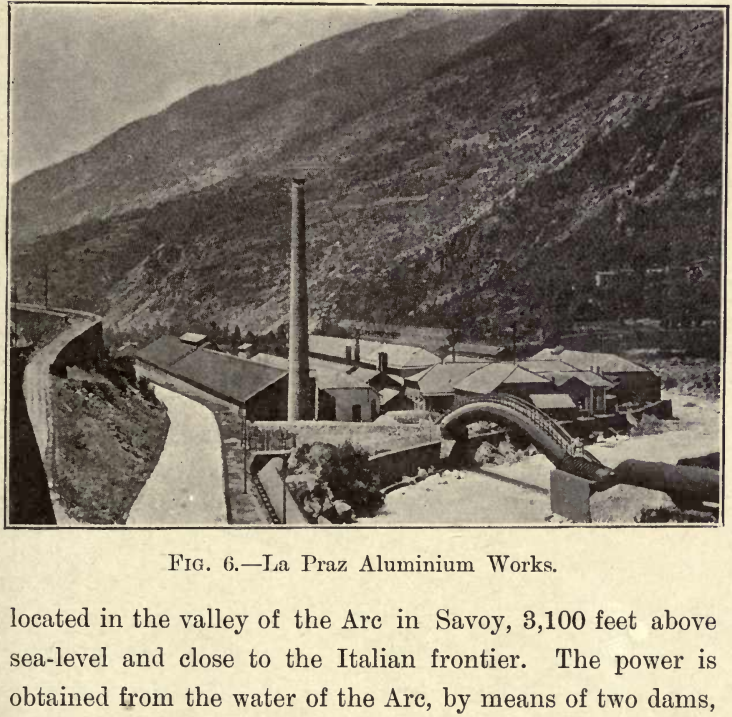


FIG. 6.—La Praz Aluminium Works.

which is 8 feet in diameter and half a mile in length, is carried over the River Arc, as shown in Fig. 7, and this forms the most novel feature of the installation.

The works at La Praz have 14,000 h.p. available, and of this total one-half is utilised by the turbine and dynamo plant. The power house contains 32 dynamos of various makes and sizes, the original units being 350 h.p. unipolar continuous-current machines, generating 7,500 amperes at 35 volts when run at a speed of 300 revolutions per minute. The later generating units are 6-pole machines, delivering 3,000 amperes at 110 volts when run at a speed of 250 revolutions per minute. The dynamos are coupled, so that currents of 9,000 amperes at 110 to 130 volts are

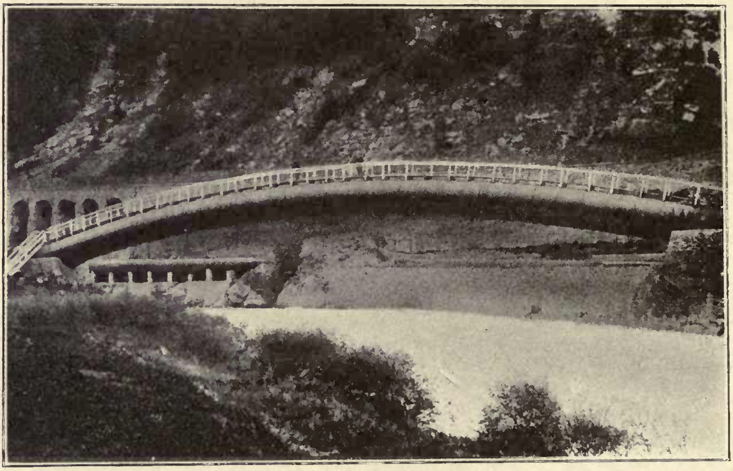


FIG. 7.—Pipe line at La Praz.

available for working the baths in which the electrolysis is carried out. Aluminium is used for all the current connections in this works.

The power available at the La Praz works has been utilised in part for production of ferro-chrome and also for steel production; consequently no figures for the aluminium production of this works, based upon the electric power generated, are reliable.

Massena, U.S.A.—The power developments at this place were planned and carried out by the St. Lawrence Power Company in the years 1899—1901. This company expended all its capital and became involved in financial difficulties before the scheme was completed, and the power plant is stated to have passed under the control of the Pittsburg Reduction Company, which operates the Hall process at Niagara Falls and at Shawinigan Falls in Canada. The power is derived from the St. Lawrence River by means of a canal three miles in length, which carries the water of this river to a point on the Grass River, where an artificial fall of 150 feet has been created. This engineering feat has been made possible by the natural configuration of the ground. The St. Lawrence and Grass Rivers run nearly parallel for a considerable distance but at different levels in this district, and therefore great facilities for power development exist.

The capacity of the generating plant at present installed is 20,050 k.w., and one-half of this total is stated to be in use in the aluminium works—erected close to the Falls. The dynamos are 2,200 k.w. machines, yielding 4,400

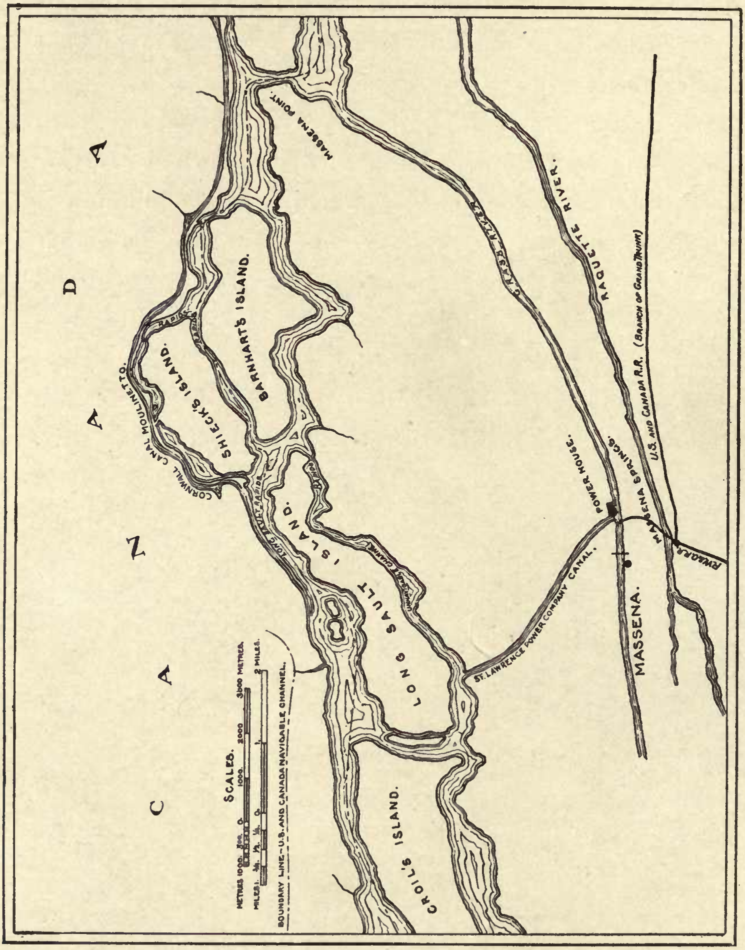


FIG. 8.—Plan of Power development, Massena.

amperes at 500 volts. Each pair of these generators is directly connected to the shafts of a single set of horizontal turbine water-wheels. Flat-bar aluminium conductors are used to convey this current of 17,600 amperes at 500 volts to the aluminium works; a feature of this use of aluminium being that the aluminium bars lie on the wood supports of the supporting poles without any insulating medium between the metal and wood. No charring of the wood has been observed at these points, and the method is certainly simple and economical.

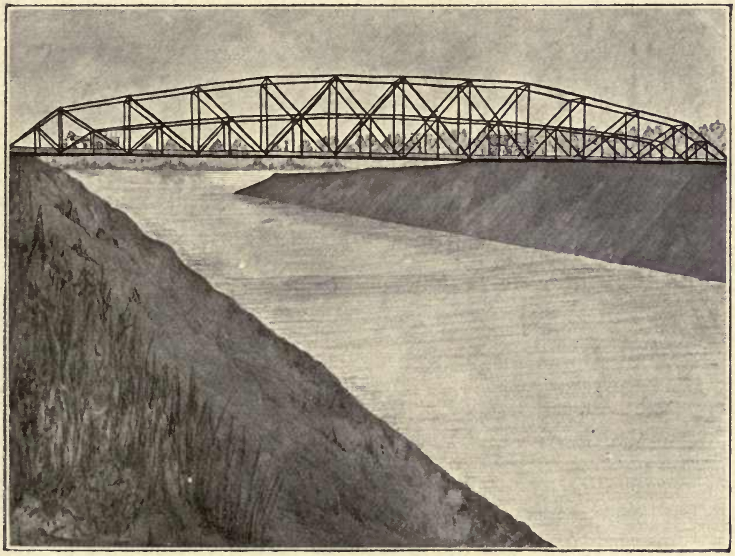


FIG. 9.—Canal Cutting at Massena.

Shawinigan Falls Works.—Shawinigan Falls is situated on the St. Maurice River in Canada, about 85 miles distant from Montreal. A fall of 145 feet is available for power purposes at this locality, and a development of 125,000 h.p.

is contemplated, of which 28,500 h.p. is now completed. The water has been diverted from the River St. Maurice just above the Falls, a canal, 900 feet long, 90 feet wide and 25 feet deep, carrying the water to the gate house of the power station. Five steel pipe lines carry the water from this gate house down to the generating station, the largest of these being 12 feet in diameter. The length of each pipe line or penstock is 400 feet. The Pittsburg Reduction Company generate their own power in a separate power house, supplied with water from two of these penstocks, the volume of water and head being equivalent to 10,000 h.p. Each turbine is of the inward-flow type, and drives directly two 1,000 k.w. continuous-current generators, mounted on the same shaft. The current is generated at 300 volts, and is transmitted to the reduction works, a distance of 700 feet, by means of aluminium conductors, in the form of rods $\frac{3}{8}$ inch in diameter.

The demand for aluminium in Europe during 1906 has been far in excess of the production, and the reduction plants are being extended in several of the works, in order to benefit by the higher prices now obtained for the metal. *The British Aluminium Company*, in addition to the development of a new water power in Switzerland, is carrying out a very large scheme on Loch Leven in Scotland, which, when completed, will add enormously to their power resources in North Britain.

The *Aluminium Industrie Ak. Gesellschaft*, of Neuhausen, are likewise developing a large power scheme on the River Navigence in Switzerland, from which it is expected that 25,000 h.p. will be derived. A new aluminium works also has been erected by an Italian company in the Valley of Pescara in Northern Italy, and is about to commence operations. In a few years, therefore, the productive capacity of the aluminium works will be more than doubled, and it will be of interest to note whether the demand shows a similar expansion.

The year 1906 has been marked by the expiration of five of the American patents granted to Hall in 1889. The Heroult patents lapsed in Europe in 1902, and the manufacture of aluminium by the electrolytic method can therefore now be carried on without the payment of patent royalties. The use of the electric current for keeping the bath in the molten state is, however, still covered in America by the Bradley Electric Furnace Patents, and these do not expire until 1909. In that country the Pittsburg Reduction Company, therefore, still possess a monopoly of the electrolytic reduction process. Owing to this fact and to the technical skill and experience demanded for the successful conduct of the electrolytic process, the manufacture of aluminium still remains a closed industry in the hands of the companies who originally introduced it in each country.[1]

---

[1] Since this paragraph was written two independent Companies have been floated in the United Kingdom, and two new works are now in course of erection.

IV. Costs.

The purity of the metal obtained by the electrolytic process has steadily improved, and the aluminium now placed upon the market contains on an average 99·6 per cent. Al. This improvement has been due to the recognition of the fact that the impurities of the raw materials used in the process collect in the reduced metal, and that the purity of the latter therefore depends upon the purity of the alumina and of the fluxes and carbon which are consumed in the bath. In the early days of the industry sufficient attention was not paid to this question, and aluminium containing 2 per cent. or more of silicon and of iron was placed on the market.

The alumina used in the electrolytic baths is obtained from a mineral called bauxite, which contains aluminium silicon and iron in the state of hydrates. Pure alumina is prepared from this by a chemical process, depending upon solution of the aluminium oxide in sodium hydrate. All of the aluminium factories now possess their own bauxite mines and refining plants for preparing the pure alumina. Hall has recently patented an electro-thermal process for treating the crude bauxite, which is expected to materially cheapen the cost of the refined alumina.

The preparation of the carbon electrodes for use in the baths also demands much care, and a subsidiary plant for making these electrodes is carried on by the British Aluminium Company, at Greenock, and by the other aluminium extraction companies at various centres in Europe and America. The cost of producing aluminium by the electrolytic process is estimated to be about 1*s.* per lb., the various items of this total being made up as follows:—

Electrical energy	1·80	pence.
Alumina	4·50	,,
Carbon electrodes	1·50	,,
Labour and management charges	1·50	,,
Stores, etc.	·75	,,
Interest, repairs and maintenance charges	1·75	,,
Total	11·80*d.*	per lb.

As the recent rise in the price of all metals, and the increased demand for aluminium, has sent up the price of the latter to 2*s.* per lb., the profits of the producing companies are now large.

The raw materials, alumina and carbon electrodes, account for one-half of the cost of production of aluminium, and any reduction in the cost must come from this source rather than from an improvement in the electrical efficiency of the process. A process which would permit crude bauxite to be used in the electrolytic baths, would, in fact, reduce the cost of aluminium production by 30 per cent. and yield a fortune to its inventor.

V. UTILISATION.

Aluminium is a white silvery metal of only 2·6 sp. gr., and most of its applications in the arts and industries are due to this extreme lightness, and to the fact that it does not tarnish on exposure to the atmosphere. Latterly, a large amount of the metal has been consumed for metallurgical purposes, its application in these cases being based upon its great affinity for oxygen, and upon the high temperature attained when the combustion (or oxidation) is allowed to occur under favourable circumstances.

Electrical Conductors.—The electrical conductivity of commercial aluminium is between 55 and 61 per cent. that of copper, when wires of equal sectional area are compared; but as the former metal is only one-third the weight of the latter, the cost of the two metals for conductors of equal carrying capacity is about equal, when copper is selling at £70 per ton in the form of wire bars, and aluminium is selling at £135 per ton, or 14½*d.* per lb. in the same form.

In the United States 56 power transmission schemes have been equipped with bare overhead aluminium conductors, and several thousands of tons of the light metal have now been employed for this purpose, especially in the Western States.

The reports which have appeared concerning the present conditions of these aluminium overhead lines in America are on the whole satisfactory, the breaks which occurred in the earlier trials being now attributed to defective metal. One of the difficulties attending the use of aluminium for this purpose—namely, that of jointing it—has been overcome by the use of a mechanical sleeve-joint, in place of brazing or soldering the two ends of the wire. The most striking illustrations of the use of aluminium for power transmission in America are between Niagara Falls and Buffalo, where an overhead line carrying 15,000 h.p. over 22 miles has been erected, and in California, where the North Yuba Power Company has an aluminium transmission line, 63 miles in length, in operation.

This use of aluminium as an electrical conductor has not progressed to a like extent in Europe; researches, carried out by Ditte in France, and by the writer, in this country, having thrown some doubts on the durability of commercial aluminium when exposed to a humid atmosphere.

Foundry and Metallurgical Uses.—The larger proportion of aluminium produced at the present time is employed for metallurgical purposes. Since the metal does not require to be specially pure for this application, a cheap grade of aluminium, containing up to 10 per cent. of iron, can be utilised.

As already pointed out, aluminium has a very high affinity for oxygen, and when mixed in the molten state with iron or steel, containing small amounts of oxide, the aluminium robs the iron of its oxygen, and rises to the surface as a light scum of Al_2O_3.

The metal is therefore used for putting into iron and steel crucibles at the moment of pouring, when castings of special fineness are required, and very large amounts have been used in the iron and steel foundries of France, Germany, and England for this purpose. From 2 to 8 oz. of aluminium are employed per ton of steel, the addition being made at

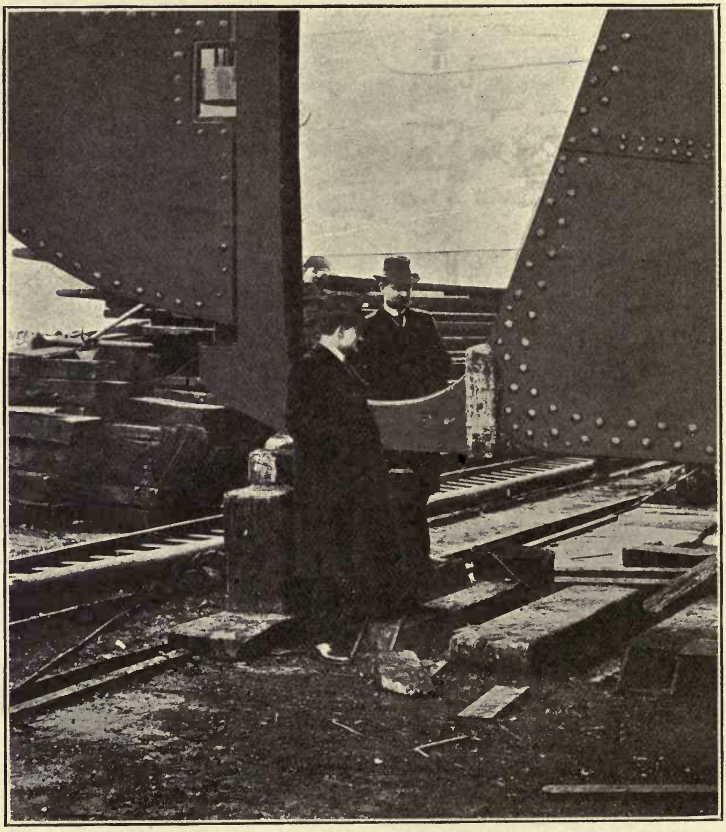


FIG. 10.—Use of "Thermit" for repairing steamer.

the moment of casting. According to M. Heroult, 2,500 tons of the lower grades of aluminium have been employed in this manner in the steel industry alone, and the consumption in this industry in France reaches 400 tons per annum. Vautin discovered, and Goldschmidt, of Essen, has worked out, the details of an application of aluminium depending on this same property. If aluminium in the form of powder be mixed with granulated ferric oxide, and the charge be ignited, an intense combustion occurs; the aluminium combines with the oxygen of the ferric oxide, while the latter is reduced to the metallic state. The heat generated during this reaction is so great that both the aluminium oxide and the iron are obtained in the molten condition at a white heat, and unless the crucible is of special quality, it also may be melted in carrying out the operation. This mixture of powdered aluminium and ferric oxide is now placed on the market under the trade name of *Thermit*, and it is being largely employed for various purposes. Separate companies have been formed in London and New York for the sale of the German *Thermit* compositions, An English company has also been floated for the manufacture and sale of a similar compound, under the trade name of *Weldite*. Litigation is at present (1906) being carried on between the English *Thermit* and *Weldite* companies respecting patent rights.[1]

Defective castings, broken mainshafts, stern posts of steamers, and broken cog-wheels can all be repaired in a very short time with *Thermit* or *Weldite*, the *modus operandi* being to form a mould round the broken parts, and to run the molten iron, obtained from a suitable charge of the aluminium compound, into this. The crucible is generally fixed above or close to the mould so that no heat may

[1] This case has ended in favour of the "Thermit" Company.

be lost in the requisite transfer of molten metal to the mould. The iron obtained from a charge of the material is at a white heat, and a true weld is obtained within the mould. The jointing of tramway rails for electrical purposes is also now carried out by aid of *Thermit* or *Weldite;* in Germany, England, and America many miles of tramway track have been welded in this manner.

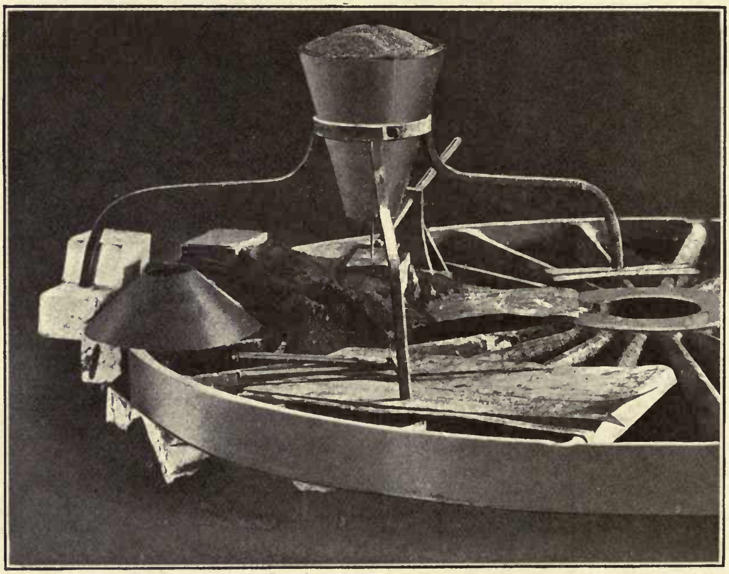


FIG. 11.—Use of "Thermit" for repairing fly-wheel.

Alloys of iron or steel with aluminium, or with other metals, can be obtained by mixing the oxides in suitable proportions, and by adding sufficient powdered aluminium to carry off the whole of the oxygen. Chromium and its alloys are now being manufactured by the Goldschmidt process at Essen; and many of the more refractory oxides have been reduced in the laboratory by similar means.

The temperature attained in the Goldschmidt process is said to approach 3,000° C., and Professor Zenghelis, of Athens, by using oxygen in place of ferric oxide, is reported to have obtained the extraordinarily high temperature of 4,000° C. by the oxidation of powdered aluminium. At this temperature platinum melts like glass, and even the temperature of the electric arc is exceeded in the crucible which contains the aluminium powder.

"*Nickel thermit*" is a new application of the "thermit" principle, this compound being employed for introducing nickel into molten cast iron. Iron containing about ½ per cent. of nickel is said to be much improved in strength, density and fineness of grain, and to have much greater resistance to chemical action. Nickel thermit is being manufactured in Germany and in the United States. *Titanium thermit:* A mixture of titanic oxide, aluminium and iron oxide, is also being used in America for improving castings of iron; the titanium being even more effective than aluminium for removing the combined oxygen and nitrogen present in the cast iron.

Printing.—A new application of aluminium which promises to become of some importance is as a substitute for stone in lithographic work. Aluminium plates with a suitably prepared surface are found to answer admirably for colour printing, and in America, England, and Germany a large number of the leading lithographic firms are now employing the new metal. The great advantages of aluminium as compared with stone, when used for colour printing, are its lightness and its adaptability to rotary presses. Both of these characteristics increase the speed with which work can be turned out. As speed means money, this use of aluminium in the printing world is certain to extend.

Alloys.—Pure aluminium is a comparatively weak metal, its breaking strain in the form of wire being about 13 tons per square inch, as compared with 30 tons for hard-drawn copper, and 35 tons for steel wire. For many purposes in which a greater degree of strength is required, aluminium is alloyed with other metals; and these alloys of the light metal have a large field of usefulness. Aluminium bronze, *Partinium* (an alloy of aluminium and tungsten), *Wolframinium* (a similar alloy), and *McAdamite* (an alloy containing aluminium, zinc and copper), are a few of these patented alloys in which strength is gained by the addition of other metals. *Magnalium* is another alloy in which aluminium is combined with magnesium, and this alloy is unique in being lighter than the aluminium. Magnalium is stated to be of great value for the construction of the metallic parts of optical and other scientific instruments, since it can be worked well in the lathe, makes fine castings, and does not tarnish.

The use of aluminium in *Heussler's alloy*, which contains copper, aluminium and manganese, and is distinctly magnetic, is another interesting application of the metal.

Zimalium is an alloy formed by adding small portions of manganese and zinc to aluminium while in the molten state. Zimalium is harder than aluminium, is easily forged, filed and drilled and has a lower conductivity than the pure metal. It is also 10 to 12 per cent. lower in price.

Miscellaneous Uses.

The use of aluminium for making small ornamental articles, such as match-boxes, photograph frames, etc., etc., is extending, and shops devoted to the sale of these goods are now a common feature of most large towns.

The lightness of the metal, its silvery appearance, and its superiority to silver as regards tarnishing, are all points in its favour when applied to this class of work.

Aluminium cooking utensils are also being widely sold, and for this purpose aluminium is stated to be superior to copper or brass. This use of the metal is rapidly extending.

Army canteen outfits and camping equipments are being constructed of the light metal, the gain in lightness being held ample compensation for the increased first cost. These canteen camp equipments have a large sale in London. Water-bottles of aluminium have also been supplied to certain regiments in the French and German armies, while aluminium helmets are receiving trial.

Aluminium has also been employed for constructing the frames of bicycles, but in this application no great degree of success has been attained, and its use for cycle manufacture is now confined to its application to frames as a paint. Some of the new flying-machines which have been experimented with during the last three years have also been constructed with aluminium rods or tubes. Here again it is questionable whether the application is wise, and whether bamboo would not be a safer material to employ. In motor-car construction aluminium alloys are reported to be much used, and, according to Heroult, this is now one of the chief outlets for the metal in France. The use of aluminium for boat-building has proved a failure, and even the much-protected deck-plates of the yachts built for contesting the American Cup match have not shown any degree of durability. This corrosion of aluminium by sea-water and by sea-air is attributed partly to the small amount of impurity which it retains, and partly to the ease with which it enters into electrolytic action when in contact with other metals.

Aluminium is being used for large castings, as for engine cylinders, etc., where great lightness is required, and where the tensile strength demanded is not high.

At a foundry in Sunderland, where a special feature has been made of this branch, seventy-five men are reported to be employed in the aluminium casting shop.

The use of aluminium in the motor-car and motor-bus industry is large and rapidly increasing. The metal is employed in thin sheets for gear cases, mudguards, and for many similar purposes, where strength is not demanded, and lightness is of some value. Cars built for some of the new tube railways in London have also been fitted with aluminium in place of wood for the inside panelling.

Cowper Coles, in London, and Schoop, in Paris, have worked out the details of a process for the autogenous welding of aluminium, which overcomes the difficulty of finding a suitable solder for the metal. By this process, aluminium sheets, rods or tubes of any thickness can be welded without any difficulty, and the joints are said to be as strong as the other parts of the metal. This method of welding will probably lead up to increased consumption of the metal in many industries, and to its use for larger articles and vessels, than have yet been manufactured from it. Another direction in which the use of aluminium is extending, is for the manufacture of pans, etc., for use in the wax-refining and jam-boiling industries, which have hitherto employed copper vessels for this purpose. The autogenous welding process is likely to extend this use of aluminium, since it will facilitate the manufacture of large vessels of this kind.

One of the latest applications of aluminium is in the manufacture of a new explosive called "Ammonal." The great heat produced by the oxidation or combustion of aluminium powder is here made use of to raise the gaseous products of combustion formed by the decomposition of ammonium nitrate, to a very high temperature. This increase of temperature intensifies the explosive effect on detonating the mixture.

VI. The Future of the Metal.

As already stated, in the first section of this chapter, the companies producing aluminium by electrolytic methods are at the present time adding greatly to the capacity of their works, and a very large increase in the output of the light metal may shortly be expected. The new works in Switzerland and Italy will soon be in full operation, and the extensions of plant at Foyers, in Scotland, and at Massena, in America, will also be completed. The Canadian Senate has appointed a select committee of five to investigate and report upon the production of alumina and aluminium within the Dominion; and it is possible that there will be some practical result from the labours of this committee, and that the manufacture will be introduced into Canada, independently of the Pittsburg Reduction Company's works at Shawinigan Falls.

Should any of the methods which have been devised for cheapening the production of aluminium prove workable, a still larger increase in production may be anticipated, for cheapened production will widen the field of application for the light metal.

The Pittsburg Reduction Company are reported to be using at the present time a process for removing the impurities of bauxite patented by Hall, which allows the bauxite to be used directly in the electrolytic bath as raw material, in place of the refined alumina. The writer is unable to state whether this method can be employed for producing the higher grades of aluminium.

Another process for cheapening the metal has been patented by A. G. Betts. This depends upon the purification of aluminium obtained from impure raw materials. The alloy containing iron, silicon, and other impurities, is used as anode material in a bath containing molten cryolite as electrolyte, and pure aluminium as cathode. These three constituents of the bath when in the molten state, float one upon the other, and the method is reported to be a practicable one. The writer, again, is unable to state whether the Betts method is in actual use on either side of the Atlantic; for all the aluminium producing companies preserve great secrecy as to the practical details of the methods employed in their works.

If aluminium could be reduced in price to 9*d*. or 1*s*. per lb., there is little doubt that it would occupy a most important place in the future. Copper and tin have advanced appreciably in value during the last five years, and the ores of these two metals, which enter so largely into the arts and industries, appear to be getting more scarce. Alumina (the oxide of aluminium), on the other hand, is one of the commonest oxides found on this earth, and forms 17 per cent. by weight of the earth's crust. Its position as one of the industrial metals of the future seems therefore assured, if cheapened methods of production are found, and alloys of strength and durability can be manufactured from it.

CHAPTER III.

BULLION REFINING AND GOLD EXTRACTION.

I. INTRODUCTION.

Bullion is the name given to alloys of the precious metals, silver and gold; the name applies also to the impure form in which these metals are sold by the mining companies, to the refiners and mint authorities in the leading cities of Europe and America. Chemical or metallurgical methods are still generally employed for separating or "parting" silver and gold from one another, and from the baser metals with which they are generally associated in the ingot form. Since the year 1895 electrolytic methods of refining bullion have been superseding gradually the older methods of work, and at the present time a very large proportion of the silver output of the world is refined by the aid of electricity. As regards gold, the older metallurgical method in which cupellation is used to remove the lead, and sulphuric acid to remove the silver, is still most widely employed, but the electrolytic method of refining is making considerable progress, and electrolytic refineries for treating gold bullion are now established in Frankfort, Hamburg, Paris, New York, Philadelphia and San Francisco.

MINING
U. OF C.
ASSOCIATION

The advantages of using electrolytic methods in bullion refining are so great that in time it would seem highly probable these methods will have entirely displaced the older chemical and metallurgical methods of parting. In place of several chemical and metallurgical processes, one operation usually suffices to dissolve the silver or gold, and to obtain it in the pure state as a deposit on the cathode. Not only is the electrolytic method simpler, therefore, than those which it may displace, but it can be carried out with much greater cleanliness, and with less risk of loss of the precious metals. The chief disadvantage is that resulting from the larger amount of gold and silver present in the electrolytic vats. The total value of this in a large refinery may run into tens of thousands of pounds, and the interest upon this capital is of course to be regarded as part of the cost of the process. It is this fact which has tended to retard the adoption of electrolytic methods for bullion refining; and the latest improvements of the methods are directed towards reducing this standing charge to the lowest possible account.

II. Historical Notes.

The principle of the electrolytic methods for refining metals is clearly set forth in Charles Watt's Patent No. 13,755 of 1851. The electrolytic procedures in use at the present day for refining silver and gold are simply developments of the ideas contained in this remarkable patent. The following extract from Watt's patent specification is of interest, in view of the important position which the electrolytic method of refining has now attained in the gold and silver industry :—

"The fourth part of my invention consists of a mode of separating metals from each other by the agency of a current or currents of electricity, and by means of vessels divided into two or more compartments separated from each other by porous partitions, such metals being at the same time freed from impurities. First, the separation of alloys of gold, silver, and copper, or of gold and silver only. If the gold alloy contains less than three-fourths of silver, or of silver and copper, I add, according to circumstances, silver, copper, zinc, lead, or any other metals which may be easily oxidised and suitable. As there is a large quantity of silver in the market which contains small but variable quantities of gold and copper, I adopt the usual method of melting a suitable quantity of this with the alloy which it is desired to purify, and the resulting compound is cast into suitable plates or bars of any convenient thickness. Sometimes the silver contains sufficient gold to render it

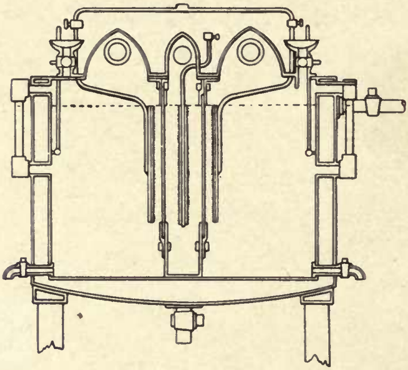


Fig. 12.—Watt's metal refining vat.

worth while to refine without the addition of any more. A solution of a salt of silver (I prefer the nitrate) is now prepared and placed in a vessel such as shown in Fig. 12, each of the compartments being filled. Sometimes for the diaphragms or porous partitions, I substitute a cell as shown in Fig. —. Or I employ a vessel to which heat can be supplied, either with partitions or diaphragms of porous material, or a cell as shown in Fig. —, without heads, funnels, or gauges, which may be of any convenient shape. The alloy in bars or plates is placed in the inner compartment and attached to one of the conductors of the current or currents of electricity, and into the outer compartment or compartments is placed a thin plate or plates of silver, connected with the other conductor of the current or currents of electricity to receive the deposited metal. The proper connections are now made between the binding screws, and the source of the electric current or currents and the action allowed to continue until the bars or plates are nearly dissolved. During this operation silver from the solution is deposited upon the receiving plate or plates, and the silver or copper (if any) of the bars or plates pass first into the solution, and then the silver is subsequently deposited upon the receiving plate or plates. If desired, the receiving plates may be taken out from time to time, and the deposited silver removed to be washed and melted in the ordinary manner. In this process only the silver and other metals oxidisable under these circumstances will pass into the solution, the gold will become detached and remain in the compartment containing the bars or plates which it is the object to purify, and when it has been collected, treated with a little nitric acid to remove any particles of silver that may adhere to or be commixed with it, will be ready to be melted in the usual way."

Moebius, a German metallurgist, in 1884 patented the silver-refining process in most general use at the present time, and greatly improved the electrolytic vat and apparatus suggested by Watt.

As the validity of patents in the electro-chemical and electro-metallurgical industries is based rather upon the novelty in construction and design of the cell and apparatus used, rather than upon novelty in principle, Watt's patent of 1851 does not invalidate this or other later patents. The Balbach Copper Company of U.S.A. have also patented a silver-refining vat, while Wohlwill, of the Hamburg Copper Refinery, has patented a process and cell for refining gold.

III. SILVER REFINING PROCESSES.

The Moebius Process.—The Moebius silver-refining vat and process is described in English Patent No. 16,584 of 1884; but modifications have been made in the design here described, as the result of practical use.

The silver bullion is used in the form of an alloy containing 950 parts of silver per 1,000 of bullion, and should contain but little copper, as otherwise this may be deposited with the silver on the cathode of the cell. As electrolyte, a dilute solution of silver nitrate is employed, containing

·12 per cent. free nitric acid, and thin sheets of pure silver are used as cathodes. The anode plates are encased in cotton bags, in which the gold slime and other insoluble impurities of the silver bullion collect; these impurities are removed at frequent intervals of time. The silver is deposited in a rough crystalline form, and by simply scraping the surface of the cathode with a wooden lath, much of the deposited silver can be detached and collected upon the floor of the vat.

The Moebius process and vat were first used at a silver works in Kansas City, U.S.A., but this installation only worked for a short time. The vat was next tried by the Pennsylvania Lead Company, whose works are at Pittsburg. This trial was commenced in 1886, and the silver refinery was gradually increased in capacity from 10,000 oz. of silver bullion per day to 40,000 oz. per day. The process was next adopted by a silver works at Pinos Altos, in Mexico, and by the St. Louis Smelting and Refining Company.

The process and vat have since been adopted in a large number of refineries situated in Europe and America. The largest European refinery is that of the *Deutche Gold- und Silber-Scheide Anstalt* at Frankfort. The following description of the Moebius plant at this works is based upon that given by Borchers in his *Elektro-Metallurgie.*

The electrolytic vats are constructed of pitch pine well tarred inside; they are 12 feet in length by 2 feet in width. Each vat is divided into seven compartments or cells by cross partitions; and in each of these compartments three anodes and four cathodes are hung. The anodes are enclosed in bags of cotton filter cloth, the cathodes hang free in the electrolyte. Four wooden laths are pivoted to a stout cross bar placed across the vat above each cathode, and these laths hang down over the cathodes. By moving the whole frame to which these cross bars are attached the laths move over the faces of the cathodes, and the crystals of deposited silver are detached and prevented from causing short circuits in the vats. A tray lined with cotton cloth is placed in the bottom of each cell to receive these detached crystals as they fall, and at intervals of 12 or 24 hours the whose contents of the cell are lifted out, and the silver is removed from this tray. The anodes are cast in the form of small plates $\frac{1}{4}$ to $\frac{1}{2}$ inch thick, with a lug for hanging them to the anode

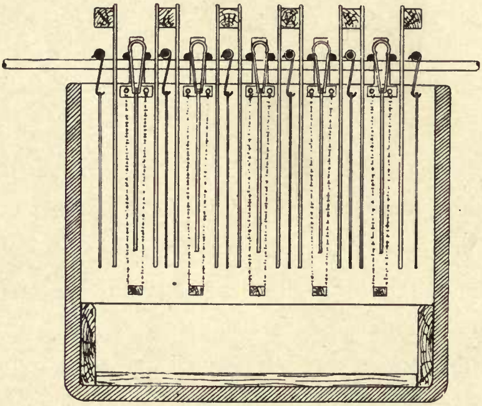


FIG. 13.--Moebius silver refining vat.

bar. Five of these anode plates are required for each anode bar, and each compartment of the vat therefore contains fifteen anode plates. The cathodes are thin sheets of rolled silver, and are connected to the current supply by means of stout copper rods to which they are soldered. The rods are hooked on to the conductors running along each side of the vat, and are in permanent connection with the negative pole of the dynamo.

The baths are filled with a dilute solution of silver nitrate containing free nitric acid in small amount. The current density may rise up to 300 amperes per square metre without danger to the purity of the deposit, provided that copper is not present in the electrolyte in appreciable amount. As the copper increases, the current density must be lowered, and the acidity of the electrolyte increased. The e.m.f. required is 1·4 to 1·5 volts per cell. Using a current of 150 amperes, each compartment of the vat described above will yield 47·5 lbs. of silver in 36 hours. As the anode plates weigh 3·3 lbs. when charged, and each compartment or cell contains fifteen of these plates, nearly the whole of the anode silver is transferred to the cathodes in the time named. The anode bags are emptied and washed out at shorter or longer intervals depending upon the amount of insoluble matter they contain, and the slime is worked up for gold and other metals. The gold in silver containing only $\frac{2}{10}$ths parts per 1,000 can be recovered by the Moebius process as described above, and this is one reason why the electrolytic silver-refining process is making considerable headway, especially in connection with the working up of the slimes obtained from the copper refineries of U.S.A. The silver obtained from the Moebius vat has a purity of $\frac{999{\cdot}5}{1000}$. As regards the size of the refineries, the *Deutsche Gold-und Silber-Scheide Anstalt* Refinery at Frankfort was originally designed for the treatment of 1,000 kgs. (2,204 lbs.) silver bullion per 24 hours. There were ten vats and seventy cells in this installation. The installation of the Moebius vat and process at the works of the St. Louis Smelting and Refining Company (shut down in 1894) was designed for the daily treatment of 933 kgs. of silver bullion, while at Pittsburg the plant was larger and was capable of dealing with 1,244 kgs. per day of 24 hours.

At the Copper Refining Works of Guggenheim Brothers at Perth Amboy, a later and improved form of the Moebius vat has been adopted. A revolving sheet cathode is placed horizontally in the vat, and is wholly immersed; the silver crystals deposited upon this are removed automatically by a scraper at one end of the vat. The anodes are placed in porous trays above the cathode, and can be examined or removed without stopping the refining process. The output of silver from the slimes of the copper refinery at this works is stated by Ulke to be 25,000 oz. per day.

The plant consists of forty-eight electrolytic vats 14 feet 3 inches × 16 feet × 7 feet deep. The electrolyte contains 4—5 per cent. copper, ·13 per cent. silver, and 10 per cent. free nitric acid. The generating plant produces 220 amperes at 90 volts, and suffices to deposit 24,000 oz. of silver per day. The original cost of the plant was £1,250, excluding the engine and dynamo, and the cost of refining is stated to be $\frac{1}{16}d.$ per oz. of silver.

At the works of the *American Smelting and Refining Company* situated at Perth Amboy, another extensive installation of the Moebius vat and process are to be found. This plant comprises 144 cells; and is operated by a dynamo of 62 k.w. capacity. Each cell takes 260 amperes at 1·75 volts, and the 144 cells are arranged in twenty-four sets of 6 cells each.

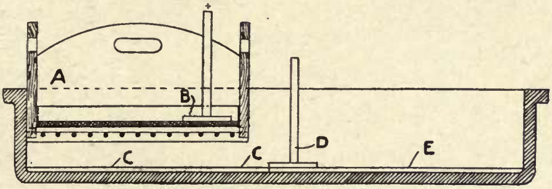


FIG. 14.—Balbach silver refining vat.

The Balbach Smelting and Refining Company's Vat and Process.—This company operates a copper-refining works at Newark, U.S.A., in which, according to Ulke, 45 tons of copper and 3,000 oz. of silver are produced per day. The silver bullion is refined in the tank shown in cross sections in Fig. 14. The bullion is cast in the form of thin square slabs. These are encased in cloth and are placed in the wooden frame marked A. This frame can be lifted out of its supports for recharging and cleaning purposes, the insoluble impurities of the anode including the gold, being found on the underside of the slabs. The electrical connection is obtained by the silver contact piece marked B—which rests on the anode slabs of Acheson graphite (c c c) ½ inch thick, fitted to the bottom of the tank. The silver contact piece D provides the electrical connection with the negative pole of the dynamo. Each tank has a cathode surface of 8 square feet, and according to Easterbrooks a current density of 20 to 25 amperes per square foot is employed. The e.m.f. required averages 3·8 volts per tank, and an average current efficiency of 93 per cent. is obtained. The power required is equal to 31·5 watt hours per oz. of fine silver produced. Most of the silver is deposited on the cathode directly beneath the anode, and must be removed at short intervals of time to prevent short circuits, due to bridging over of the gap by silver crystals. According to Easterbrooks, the Balbach process and vat for silver refining has a higher nitric acid consumption than the Moebius process and vat owing to the liberation of nitrogen oxides at the anode, but the amount of silver retained in the vat and electrolyte is lower than with the latter. The actual working costs are therefore higher, while the interest charges are lower than with the Moebius process and vat.

The Balbach process and vat are not in use, so far as the author is aware, in any other refinery than the Balbach Company's Refinery at Newark.

The chief disadvantages of both the Balbach and Moebius vats are due to the crystalline form in which the silver is deposited at the cathode. This necessitates a wider gap between anode and cathode, and a higher e.m.f. than would otherwise be required for working the process. Experiments are therefore now being made with additions of gelatine to the electrolyte, with the idea of obtaining a smooth and coherent deposit of silver, with a current density of 10 amperes per square foot. Agitation of the electrolyte also assists in the attainment of smooth and coherent deposits, as in the case of copper. Easterbrooks has proposed a vat and process based upon these principles, with a system of circulation and electrode arrangement similar to that used in the multiple system of copper refining.

The Dietzel Silver Refining Process.—This process for separating copper from the silver and gold in poor alloys, is worked at the Gold u. Silber-Scheide Anstalt, at Pforzheim, in Germany. In place of using a dilute solution of nitrate of silver as solvent, a solution of copper nitrate is employed, in a vat divided by a diaphragm into two compartments. This solution takes up silver and copper in the anode compartment of the vat. It is then removed from the anode chamber to another vessel, in which it is treated with copper filings and cuttings. The copper here displaces the silver by chemical action, and the resultant solution of copper nitrate, containing excess of nitric acid, is then allowed to pass through the cathode compartment of the electrolytic vat and the copper is deposited upon pure copper sheet cathodes. The gold is obtained in the insoluble residue or slimes from the anodic compartment of the vat; while the silver is obtained as a dark deposit in the vessel containing copper filings and cuttings. Under favourable conditions the value of the recovered copper pays all the costs of the process.

The Betts Process.—A. G. Betts, of Troy (New York), has suggested and patented a new method for separating or parting silver from lead, bismuth, antimony and other impurities. The electrolyte contains 4 per cent. of silver in the form of silver methyl sulphate, and 4 per cent. of methyl sulphuric acid; while an addition of carbon disulphide is found to harden the deposit of silver. A current density of 10 amperes per square foot of cathode surface is employed. The deposited silver is hard and solid, and is free from the "treeing" which causes much difficulty in operating the Moebius and Balbach processes of refining.

The process is covered by U.S.A. Patent No. 795,887 of 1904; and a full description of it will be found in the "Transactions of the American Electro-Chemical Society." Vol. VIII. p. 121.

As regards the extent to which electrolytic methods are used in the production of silver, no less than 74,000 oz. of silver are produced daily (equivalent to 27,000,000 oz. per year) by the copper refineries of the United States, and in the majority of these refineries electrolytic methods are employed for parting this silver from the gold with which it is alloyed. The slimes from the silver parting process in these same refineries, according to Ulke, yield 948 oz. of gold per day or 346,000 oz. per year. The total value of these two "impurities" recovered annually from the slimes of the copper refineries of the United States is over £4,000,000. In Germany, the Norddeutsche Affinerie produces 100,000 kgs. of silver per year, while the Mansfelder Copper Company produces 4,000 to 5,000 kgs. of silver from the copper slimes of the copper-refining vats. The Stadtberger Hutte at Niedermarsburg is another copper company recovering silver from the slimes of the copper vats.

There are several refineries of this kind in the United Kingdom, but no details of the plant can be published.

IV. Gold Refining Process.

The Wohlwill Process.—This process for separating gold from silver and from the metals of the platinum group was not patented by Wohlwill in the United Kingdom until 1896, although the details of the process had been worked out in the works of the Norddeutsche Affinerie at Hamburg, in the years 1878—1881; and a bar of fine gold testing $\frac{1000}{1000}$, refined by this method, had been shown at the Paris Exhibition of 1881.

The Wohlwill process depends upon the use of the impure gold as anode in a cell containing an acidified solution of gold chloride, using cathodes of thin sheet gold. When an electric current is passed though this solution the gold and platinum pass into solution as chlorides, while the silver is converted into the insoluble chloride, and collects with some gold, as anode slime upon the bottom of the cell. The electrolyte must be made up with distilled water and must contain 25 to 30 grams of gold chloride, with 40 cubic centimetres of fuming hydrochloric acid per litre. The temperature must be maintained at from 60° to 70° C. during the refining process. The anodes of gold bullion are made 4 m.m. ($\frac{1}{6}$ inch) in thickness, and may contain up to 15 per cent. of silver. Using an initial current density of 38 amperes per square foot, rising to a maximum of 190 amperes per square foot, 80 per cent. of the anode gold will have passed into solution and have been re-deposited in 24 hours; 10 per cent. will be found in the anode slimes; and 10 per cent. will still hang together as anode and must be carefully removed from the vat, washed and re-cast. The distance separating anodes and cathodes must be 3 centimetres ($1\frac{1}{4}$ inches). Earthenware or porcelain must be employed as material for the cells. The anode slime is removed from the bottom of the cell at short intervals of time, and is submitted to chemical treatment. When the gold bullion is very rich in silver, an automatic mechanical device is used for removing the silver chloride from the face of the anode, to which it has a tendency to cling. The platinum and other impurities which collect in the electrolyte can be allowed to accumulate for a considerable period, as their presence tends to increase the density of the gold deposit on the cathode. Palladium is the only metal which has an opposite effect, and the amount of this in solution must not be allowed to exceed 5 per cent. The gold is deposited on the cathode in coarse crystalline particles which can be easily detached. The crystals are washed with distilled water and are then melted and cast into bars. The average fineness of the bar-gold is 999·8. The gold depositing vats of the Frankfürter Gold-und Silber-Scheide Anstalt, at Hamburg, suffice for the deposition of 75 kgs. fine gold per 24 hours, and cover only 64 square feet of ground space. The e.m.f. required to maintain a current density of 95 amperes per square foot is 1 volt. The theory of the Wohlwill process is less simple than it appears to be, for the electrolyte contains both aurous and auric chloride (AuCl and $AuCl_3$), and the presence of gold in the anode slime is ascribed to the following reaction:—

$$3AuCl = AuCl_3 + 2Au.$$

The use of an excess of hydrochloric acid, and of a rather high current density at the anode, are necessary in order to reduce to a minimum the formation of aurous chloride, and of the finely-divided metallic gold resulting from the secondary reaction. On the other hand, no gold is dissolved at the anode, if the solution be kept neutral.

The advantages claimed for the Wohlwill process of gold refining are its greater economy and convenience, as compared with the older sulphuric acid and aqua regia methods of separating gold from silver and platinum. In the electrolytic method the acid solution is used over and over again—in the older method this was impossible. By the Wohlwill process the gold is dissolved from the anode and deposited at the cathode in one operation; by the older method many steps marked this stage of the refining process. Finally, though gold bullion contains only small quantities of platinum and palladium, the accumulation of a few months' refining work may prove of considerable value. With a daily treatment of 75 kgs. of gold bullion containing only 1 in 1,000 of platinum, 100 days' work will yield 7·5 kgs. of the latter metal, and this, at the present market price, is worth £1,800.

The Wohlwill electrolytic refining process is now in use at the following refineries : Frankfort, Hamburg, Paris, New York, Philadelphia, and San Francisco. At the U.S.A. Mint, at Philadelphia, the process is employed for gold bullion, containing 40 to 60 parts per 1,000, of silver, platinum, copper, lead, etc. The plant consists of seven cells, measuring 32 × 27 × 20 centimetres, and containing, as electrolyte, a solution of $AuCl_3 = 30$ grams gold chloride per litre. The output is 5,000 oz. of fine gold per week. A current of 100 amperes at 4·5 to 5 volts is used for each cell.

V. Gold Extraction Processes.

The Siemens and Halske Process.—This process is a modification of the well-known McArthur-Forest cyanide process, and it received extended trials in the Transvaal, South Africa, in the years 1893—1896.

The great difficulty in working out a successful electrolytic process for this operation, was due to the very diluted state in which the cyanide solution contained the soluble gold cyanide. It was found impossible to remove the whole of the gold from these solutions, without multiplying unduly the number of vats and electrodes, and the plan adopted was to deposit only a portion of the gold, and to return the electrolyte to the vats for a fresh leaching operation.

Long wooden tanks 20 feet × 8 feet wide × 4 feet deep were employed, with the electrodes placed very close together in such a manner that the cyanide solution was compelled to take a zigzag course along the tank. The anodes were of iron, 7 × 3 feet × ⅛ inch thick; the cathodes of thin sheet lead. Hundreds of pairs of electrodes were employed, the iron anodes being encased in coarse canvas bags to retain the prussian blue, and the lead cathodes being held in wooden frames. Each vat contained seventy-one pairs of electrodes spaced 1½ inches apart, and one of the earliest plants had four of these vats, or 284 electrodes in operation. An e.m.f. of 4 volts was sufficient to yield a current of ·05 amperes per square foot of anode surface. The vats were covered and locked, and only required opening once a month for removal of the gold-coated lead cathodes and insertion of new cathodes in the wooden frames. The lead cathodes were washed and submitted to cupellation for the final recovery of the gold.

The costs of operating the electrolytic process were stated to be lower than the older zinc-box recovery process; and were estimated at 33*d.* per ton of tailings for the whole cyanide process.

Improvements have been made in the zinc recovery process, however, which have rendered it the more economical, and the writer believes that the electrolytic recovery process of Siemens and Halske has failed to establish itself permanently on the Rand Gold Fields.

The Greenawalt Process.—An extraction process which makes use of electricity for dissolving and recovering gold and silver, has been developed by Greenawalt in connection with the Wallstreet Mills of the Nancy Gold Mining Company, Colorado. The ore is first crushed and roasted, and is then leached with a solution containing free chlorine and bromine. This leaching solution is produced by electrolysis of a strong brine, containing sodium bromide in addition to sodium chloride. The solution of gold and silver obtained in this way from the leaching vats is then passed through depositing or precipitation vats, where an electric current is again used to recover the precious metals. Lead plates spaced 1½ inches apart are used as electrodes in the depositing tanks; one half of these being connected to the negative conductor and the other half to the positive conductor. A current density of $\frac{1}{36}$th ampere per square foot is employed.

The electrolyte passes through the tank in a zigzag manner. The deposit on the cathodes is obtained in the form of a slimy black powder, and this is collected and refined by the usual methods.

The Greenawalt electro-chlorination process for extracting gold from suitable ores is being exploited by the Wallstreet Gold Extraction Company of Wallstreet, Boulder County, Colorado.

CHAPTER IV.

CALCIUM CARBIDE AND ACETYLENE GAS.

I. HISTORICAL INTRODUCTION.

THE fact that the element carbon could enter into combination with a metal was discovered by Davy one hundred years ago, for in the year 1808 that noted English chemist produced the compound potassium carbide, only one year after his discovery and separation of the metal potassium. Nearly thirty years later, or in 1836, his cousin Robert discovered the analogous compound *calcium carbide*, and Wohler in 1862 repeated Davy's experiments and studied the properties of the new compound. The formation of calcium carbide when calcium oxide and carbon were heated to a very high temperature, and the production of acetylene gas by decomposition of this compound when treated with water, were therefore known nearly half a century ago, but not until the closing years of the century did these discoveries become of industrial significance or importance. In the years 1892—1893 Moissan, the famous French chemist, in Paris, and Willson, an American electro-metallurgist, at Spray, North Carolina, practically re-discovered the compound calcium carbide as a result of experiments with the electric furnace.

Willson appears to have been the first to re-discover the compound, during experimental work upon the production of aluminium, by means of the reducing action of metallic calcium upon alumina (Al_2O_3). The metallic calcium was to be obtained by heating lime with carbon in an electric furnace, and it was when carrying out this portion of the process that Willson accidentally obtained calcium carbide. A sample of the product was taken from the furnace and thrown into water, and the experimenters were surprised to find that it gave off a gas which could be ignited, and burned with a luminous but sooty flame. There was some delay on the part of Willson and his assistants in ascertaining the composition of this furnace product, and of the gas liberated by it on contact with water. Although Willson's experiments dated from May, 1892, Moissan, of Paris, who carried out his investigations in a more scientific manner, deservedly obtained most credit for the re-discovery of calcium carbide, and for pointing out the possibilities of this compound as the source of a new illuminant.

The first patents were applied for in Germany, France, America, and the United Kingdom in 1894, the claim being for a process whereby calcium carbide could be produced by heating lime and carbon to a very high temperature in the electric furnace.

The French and German patents were applied for in the name of Bullier—Moissan's assistant at that date—and the American and English patents were granted in the name of Willson, the American inventor. Companies were immediately formed in England, France, Germany, and America for exploiting the manufacture of the new product; and the production of carbide was commenced in the United Kingdom at Foyers, in Scotland, in October, 1896.

The patent rights of the Willson process for the United Kingdom had been purchased by a company named "The Acetylene Illuminating Company," formed in May, 1895, and this company arranged with the British Aluminium Company to utilise part of the power available at Foyers for the manufacture of the product, calcium carbide.

The new compound and its use for production of acetylene gas aroused great interest during the succeeding three years, and very exaggerated claims were made for the place this gas was to take, as a rival of coal gas and electricity. Companies for the manufacture of calcium carbide and for the manufacture of acetylene generators were floated in absurd numbers, and patents for acetylene generators were granted by the British Patent Office with ridiculous frequency. In the years 1896—1899 over 650 patents for acetylene generators *were granted* in the United Kingdom alone, and during the same period fifteen companies, with an aggregate capital of nearly half a million sterling, were floated in London for carrying on the manufacture of calcium carbide or for exploiting new forms of acetylene apparatus.

When it is remembered that the principles of acetylene generation simply involved slow or quick contact of calcium carbide with water, and that the special arrangements adopted in most of the patented generators for bringing this about had been in use for years in chemical laboratories for generation of SH_2 and other gases, the value of the majority of these patents, and of the shares in the companies formed to work them, may be imagined. Even had the calcium carbide industry developed in this and other countries to the extent which company promoters and patentees had expected at this date (1896—1898), the greater number of these so-called patents would have been found to be invalid and of no commercial or industrial value.

However, for three years the calcium carbide industry enjoyed a brief spell of prosperity, both in this country and on the continent of Europe. The manufacture of carbide was commenced near Bradford, at Wood Green near London, at Ingleton in North Lancashire, and at Cradley Heath near Birmingham, and the companies for acetylene gas generation increased rapidly in number. The speculation was not confined to the United Kingdom. In France and Switzerland a similar unhealthy and over-capitalised development occurred in the new industries. Water powers in every district of France were feverishly developed for manufacture of calcium carbide, and in the year 1899 no fewer than sixty-eight carbide works were in operation in Europe, with sixteen additional ones in course of erection.

A collapse was bond to follow this unwise "boom" in the new manufacture. Acetylene gas proved less satisfactory and less economical as an illuminant than was expected, and several fatal generator accidents shook public confidence in the safety of the new gas. The present writer, in an article published in October, 1897, and others at that date foretold the probable results of this over-capitalisation of the new industries, and of the exaggerated claims made for acetylene gas as an illuminant.

In the years 1899—1900, the collapse came with startling suddenness, and of the original companies formed to promote the new manufactures the greater number have ceased to exist, while of the remainder hardly one exists to-day that has not passed through financial liquidation and reorganisation.

To add to the troubles of the original companies formed to carry on the manufacture of carbide, the validity of the foreign patents granted to Bullier and Willson began to be questioned, and at the present date only the patents granted to Bullier in France have been left intact by legal decisions. In Austria, Germany, Switzerland, and the United Kingdom, the patents have been declared invalid, and the manufacture of calcium carbide is therefore open to all who care to take it up. In the United States, the owners of the Bradley Electric Furnace Patents, and not the Willson Company, control the manufacture of carbide and other electric-furnace products, and receive a royalty on each ton of carbide made.

II. The Carbide Furnace and Process.

The production of calcium carbide by heating lime and coke together, in the absence of air, to a high temperature, is in principle an ordinary chemical process ; and electricity is simply made use of because the temperature required can be most easily attained in this way.

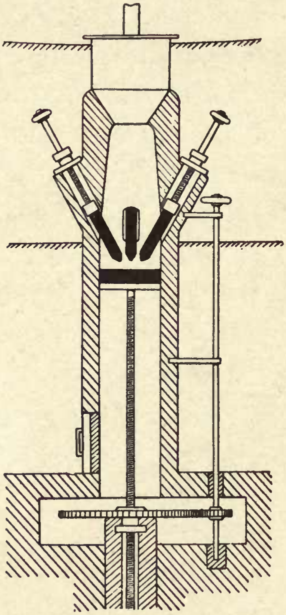


Fig. 15.—Continuous carbide furnace (Memmo type).

The reaction expressed in chemical notation is as follows:—

$$CaO + C_3 = CaC_2 + CO.$$

The carbonic oxide is given off as gas during the reaction, and the manufacture therefore demands furnace rooms or buildings with good ventilation. The precise temperature required for the reaction is not yet settled, but it is over 2,000° C., and assuming it to be 3,300° C., it has been calculated that the theoretical yield of carbide, for a furnace worked continuously, would be 35¾ cwts. per e.h.p. year. The average yield has been much below this figure. The best yield in well-managed works is 2 tons per k.w. year ; but the average has been about 25 cwts. carbide per e.h.p. year.

Two types of furnace have been employed, the continuous and the intermittent. Either direct or alternating current can be employed, since the electricity is used solely as a heating agent. Fig. 15 is a sectional elevation of a furnace designed for continuous working, and Fig. 16 is a sectional elevation of a furnace for intermittent working by three-phase current. These furnaces are the design of R. Memmo, an Italian engineer, and have been used at the carbide works at San Marcello d'Aosta, in Northern Italy. The first furnace is cylindrical in shape, and is built of refractory bricks. The three carbons which convey the three-phase current are inclined to the base plate at an angle of 60° as shown, and are controlled by threaded rods, each operated by a small hand-wheel. The base is formed of a cast iron plate covered with graphite, which may be raised or lowered the whole height of the furnace by means of a screw operated by gear and pinion, as shown in the figure. The raw materials are charged through the metal funnel provided at the top of the furnace, and as the carbide is formed under the intense heating effect of the three arcs, the base plate is caused slowly to descend, and the column fills with carbide, molten at the top but comparatively cold at the bottom. The product is removed continuously by the door shown on the left at the foot of the column. The chief difficulty met with in the operation of this furnace, is to regulate the feed of raw materials and removal of finished carbide to the heating capacity of the arcs. Later forms of continuous carbide furnaces have been designed with moveable vertical electrodes, in place of a moveable base plate. Others have been constructed upon the rotary plan, with the arcs playing on one portion of the periphery of a revolving wheel, upon which the raw materials are fed, and from which the carbide is removed at another portion of its circuit. The last-named plan is the most efficient, since the regulation of the period during which the materials are submitted to the heating action of the arc is easily varied by increasing or decreasing the speed of rotation, while the charging of raw materials and removal of the finished carbide is also simple and effective. The chief difficulty in this form of furnace is the exclusion of the air. Air causes loss in two ways—first by combustion of the carbon electrodes, and secondly by its action upon the carbide and formation of calcium cyanamide. For this reason the intermittent or closed type of furnace, in which the carbide is allowed to cool in an inert atmosphere, is still used in many works. Willson's and Moissan's original furnaces for carbide production were of this type, and Fig. 16 shows the modified form, adapted for three-phase current. The internal walls of this furnace are built of refractory bricks, while the outside walls are of ordinary

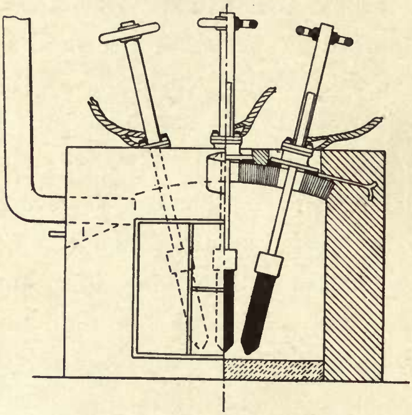


Fig. 16.—Intermittent carbide furnace (Memmo type).

bricks. The bottom of the furnace is built of compressed blocks of magnesium oxide or of lime. In actual working the side walls of the furnace are protected by the crust of carbide which is allowed to remain for this purpose. The furnace is filled with the necessary proportions of lime and coke before starting the current, and the carbons are gradually drawn up to the top of the furnace as the interior fills with the molten carbide. When the whole of the charge of raw materials has been converted into carbide the current is cut off, and the furnace is allowed to cool before the door is taken off and the carbide is withdrawn. A further advantage of this type of furnace is that the direct heating can be stopped before the conversion into carbide is quite finished, and the final stage of the reaction can be allowed to complete itself with the aid of the heat stored in the furnace walls and materials contained within it. This gain to some extent compensates for the loss of heat due to radiation and to the intermittent character of the furnace operation. In the latest types of the intermittent furnaces attempts have been made to minimise this loss by removing the block of carbide as soon as it is sufficiently set, and by starting the furnace again while the walls and base still contain a large reserve of heat. Pre-heating of the raw materials by the exit gases of the carbide furnace has also been adopted.

Numerous modifications of these two types of furnace have been devised and patented, but these modifications relate chiefly to the minor details of furnace construction; and the leading principles of carbide furnace design described above are now generally accepted by all manufacturers.

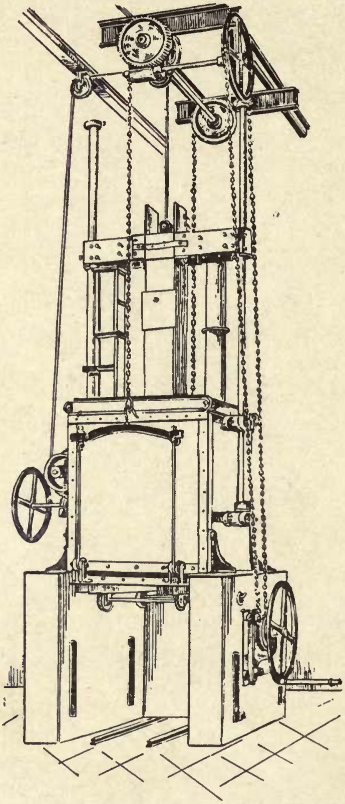


FIG. 17.—Intermittent carbide furnace (Carlsen type).

III. PRODUCTION.

Production of Carbide in the United Kingdom.—The present position of the industry in the United Kingdom is as follows:—

The Acetylene Illuminating Company have unsuccessfully sought to maintain the validity of the Willson patents granted in 1894, and purchased by them in 1895. Though the case was tried three times and carried in 1904 to the final Court of Appeal in the House of Lords, the decision was twice given in favour of the defendant company—the United Alkali Company, of Liverpool. The manufacture of carbide at Foyers, owing to this and other reasons, has been finally stopped, and the Acetylene Company now acts as a distributing agent for carbide obtained from various sources.

The manufacture at Foyers, which commenced in 1896, was at first carried on by the Acetylene Illuminating Company, who rented electric power from the British Aluminium Company. According to Professor Vivian Lewes, the output in the early days of the industry at this factory was ·40 lb. of carbide per e.h.p. hour, or 9·6 lbs. per e.h.p. day. The furnaces used at this date were of the intermittent type, and absorbed 1,200 amperes at 60 to 70 volts. The yield obtained from the furnaces at Foyers is believed to have improved, as more experience was gained in controlling the temperature of the reaction, but it never approached the 20·0 lbs. per e.h.p. day reported to have been obtained at Spray in the experimental trials of the Willson furnace and process. Willson calculated from these early experiments that 1,000 kgs. of carbide (equal to 1 metric ton of 2,204 lbs.) could be produced with an energy consumption of 2,400 e.h.p. hours, and that 1,000 kgs. of lime and 533 kgs. of coke would yield this quantity of carbide.

Experience proved that this estimate was too low in every respect, and many of the failures to make the manufacture of carbide a financial success in Europe have been due to this original under-estimate of the cost of production.

In 1899 the British Aluminium Company took over the control of the carbide manufacture at Foyers, and operated the Willson furnaces and process under an arrangement with the Acetylene Illuminating Company, which contracted to take and sell the whole of the product at a stipulated price. This arrangement lasted till 1904, when the manufacture of carbide at Foyers finally ceased.

No details of any value can be given concerning the other companies or works for carbide production which were erected and operated in the United Kingdom during the years of "boom," 1896—1899.

As already stated, the works were erected at Bradford, Ingleton, Wood Green, near London, and Cradley Heath—and all of these have now ceased production. The plant at Cradley Heath was the largest of these installations, and comprised a 300 k.w. generator, and eight carbide furnaces.

The United Alkali Company of Liverpool, the successful defendants in the patent infringement case with the Acetylene Illuminating Company, also operated a carbide plant at Widnes, for a few months, about the year 1900, The plant was designed by and operated under the control of Mr. J. Bastick, but it was not kept in constant operation, owing to the high cost of electrical power at this place. With the exception of Ingleton, all these works were dependent upon steam power for generation of the electrical energy required. The Ingleton works, where the Parker furnace was employed, were burned down in 1904, and have not been rebuilt.

The only carbide factory known to the writer, now at work in the United Kingdom, is situated at Askeaton, Co. Limerick, in Ireland, where water power is available. A mill on the River Deel at this place has been rebuilt, and a 400 h.p. turbine, coupled to a 300 k.w. generator, has been installed. The electric furnaces are of the Bastick design, and each takes 3,000 amperes at 100 volts when in full work.

No figures can be given for the production of carbide at this works, but it is necessarily only small, and at no time in the history of the industry has the output of carbide by all of the works erected in the United Kingdom amounted to 3,000 tons per annum.

This restricted development of the carbide industry in the United Kingdom has been partly due to the lack of certainty as to the validity or invalidity of the Willson patent rights, but chiefly to the cost of electricity when generated by coal and steam power, and the consequent inability to manufacture carbide under these conditions as cheaply as it could be manufactured at, and imported from, water-power centres abroad.

At the present time the bulk of the carbide used in the United Kingdom is imported from Sweden and Norway. There seems little chance that the industry can be revived in the United Kingdom, unless the waste gases of blast furnaces are utilised for generating electric power at very low cost, by means of huge gas engines coupled to dynamos.

A somewhat similar scheme for carbide production is now in course of development in Yorkshire in connection with the generating station of the Yorkshire Power Company.

The price of carbide in the United Kingdom has fallen considerably since the early days of the industry, and the following quotations from the price lists issued in various years by the Acetylene Illuminating Company, are instructive.

The early estimates of cost by Willson at Spray, were, it must be remembered, £6 10*s.* to £7 5*s.* per ton, while estimates based on experiments at Meran in Austria, yielded a total cost of £7 5*s.* per ton.

The prices quoted by the Acetylene Company for Foyers carbide in 1896 was, however, £28 per ton, with discounts varying from 10 per cent. to 25 per cent.

In 1900 the price quoted had dropped to 19*s.* per cwt., or £19 per ton, while in November, 1903, a further reduction in price to 14*s.* per cwt. or £14 per ton had occurred. These prices were f.o.r. at Foyers. The 1906 quotation of the Acetylene Company for imported carbide, at ports in the United Kingdom, was 14*s.* per ton.

Production of Carbide Abroad.—As regards the manufacture of calcium carbide in Europe and America, according to the most recent and reliable returns, there are now between sixty and seventy works engaged in the industry, and the aggregate production of carbide amounts to between 90,000 and 100,000 tons per annum. The United States, Italy and France head the list of producing countries, and are also the largest consumers of carbide for acetylene generation.

In France, the industry is controlled by a syndicate with

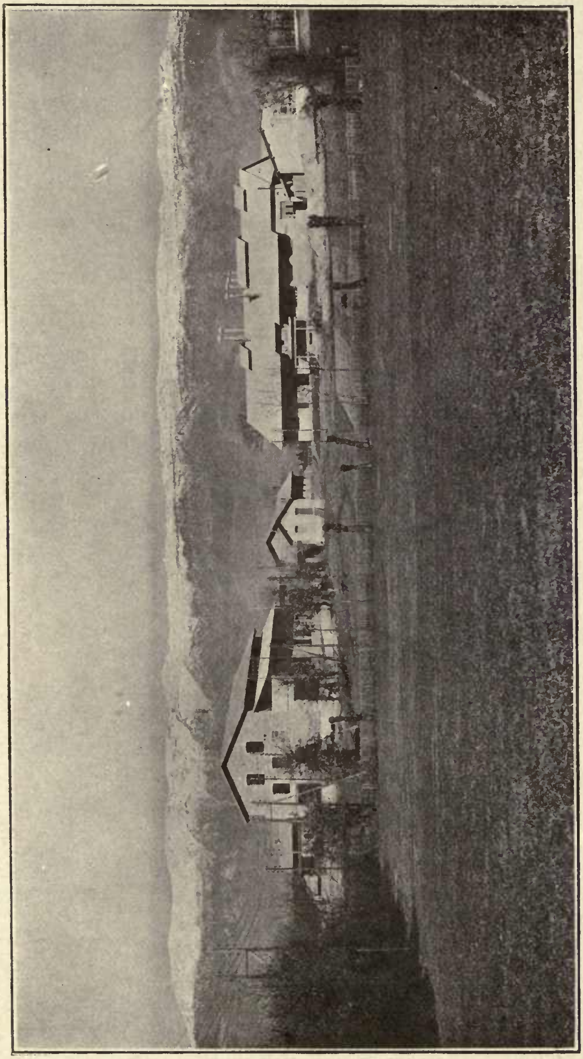


FIG. 18.—The Volta Carbide Works at Vernier, near Geneva.

head quarters in Paris, and this exercises a close watch over output and price. Eleven works are reported to be still operating, situated around the following centres of cheap water power: Bellegarde, Grenoble, Nice and Toulouse. The estimated output of these works in 1906 was 24,000 tons, while the annual consumption in France is about 15,000 tons.

Germany is dependent upon Switzerland, Austria and Norway for two-thirds of its supply of carbide, only 8,000 tons being produced at home, while 16,000 tons are imported. In the United States the production of carbide is estimated to amount to 25,000 tons per annum, the Union Carbide Company, with works at Niagara Falls, being the chief producers. A large new factory designed for utilisation of 10,000 h.p. is, however, now being erected at Duluth.

IV. UTILISATION.

1. THE ACETYLENE INDUSTRY.

When calcium carbide is brought into contact with water, a violent reaction occurs with the production of heat, and acetylene gas is liberated according to the following chemical equation:—

$$CaC_2 + 2H_2O = Ca(OH)_2 + C_2H_2.$$

The hydrate of lime partly dissolves in the water, and partly separates as a flocculent precipitate, while the gas escapes. Acetylene is a colourless gas, with a sp. gr.

of ·90, and therefore it is much heavier than coal gas, the sp. gr. of which is only ·40. Acetylene possesses a most nauseating and penetrating odour.

Acetylene generating machines are of two types, according to the manner in which the calcium carbide and water are brought into contact. The carbide-to-water generators are the better in principle, since the large volume of water used in these serves to keep down the temperature, and to wash the liberated gas. The other type of generator is, however, employed, in those cases where it is not practicable to use a vessel of large capacity for holding water.

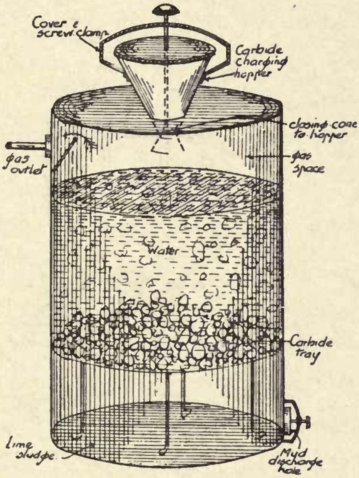


Fig. 19.—Acetylene generator (simplest type).

Fig. 19 shows a simple form of carbide-to-water generator. In the early days of the industry, acetylene generators with automatic feeding devices were patented by the score, but experience has proved that it is safer to dispense with complicated devices of this kind, and to feed the carbide into the generator by hand, as in the generator shown in Fig. 19. This method of generating acetylene of course renders it necessary that the generator should be provided with a gas-holder for storing the gas. A purifying plant is also necessary if the acetylene is to be used for indoor illumination, since the gas contains, as a rule, several very dangerous gaseous impurities.

As regards the extent of the acetylene industry, although the hopes of inventors and company promoters have not been realised, and although the unwise company promotion and financial speculation of the earlier days have hampered the growth of the acetylene industry, considerable progress is to be recorded. As regards actual installations, the following is a list of the places in the United Kingdom where central acetylene gasworks have been erected and operated for the supply of the gas for public and private lighting purposes:—

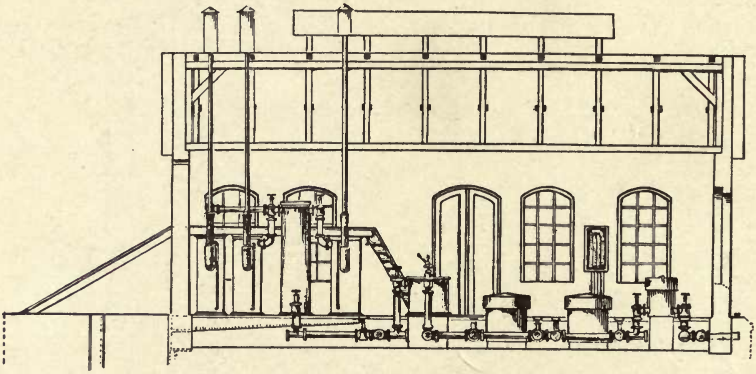


FIG. 20.—Central acetylene generating plant at Oliva, Germany.

Aberfoyle, Arklow, Craven Arms, Dunfanaghy, Ederclevry, Gost, Hawes, Hillesley, Hunmanby, Lee-on-Solent, Neagh, Rootoy, St. Michael's, Tandragee, West Swinton, and Wilhorn.

The writer is not in a position to state, however, that all of these installations are still in operation.

The average price charged for the gas to consumers is stated to be 55*s.* 3*d.* per thousand cubic feet, as compared with 2*s.* 8*d.*, the average for coal gas in the larger English cities and towns.

As regards private installations of acetylene gas, one firm alone is stated to have erected generators in 100 residences, in nine churches, and in several manufacturing establishments, while a large number of factories in out-of-the-way localities and of gentlemen's country houses have been lighted by the new illuminant.

Portable acetylene flare-lights have also been used on the Manchester Ship Canal for night work.

The following details of the installation of acetylene lighting at Hunmanby are typical of many similar village generating and distributing plants. Hunmanby is a village of 1,000 inhabitants, situated on the east coast of Yorkshire. Acetylene lighting was first adopted in 1897, under the auspices of the Electroid Gas Company. This installation was not, however, successful owing to leakage in the gas mains and to a mistaken admixture of air with the gas supplied, and it was not until November, 1900, that a successful supply was inaugurated by the Acetylene Illuminating Company. Three generators of the non-automatic "Ideal" type were installed, with the necessary purifying plant and gas meters, also a new gas main and distributing branches. The distributing main was of wrought-iron barrel pipe, ranging from 2 inches down to $\frac{3}{4}$ inch in diameter. The generators each required a 20 lb. charge of carbide and produced 100 cubic feet of acetylene. The gas-holder was placed behind the generator house, and had a capacity of over 500 cubic feet, which represented the normal winter 24 hours' demand at the time the plant was installed. The charge made for the gas at Hunmanby was 60*s*. per 1,000 cubic feet, with a meter rate of 1*s*. per quarter. This charge was stated to be equivalent to coal gas at 4*s*. per 1,000 cubic feet, but since the fall in the price of carbide the charge for acetylene has probably been reduced.

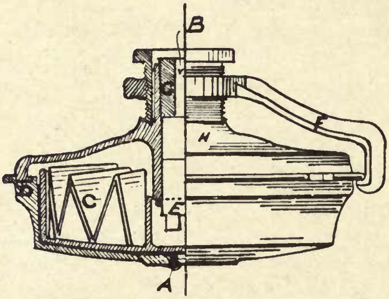


Fig. 21.—The Lewis acetylene lamp.

The use of acetylene for road vehicles is also progressing, and in 1906 over 200 of the London omnibuses have been lighted by acetylene with successful results. The earlier trials in 1901—1902 with a lamp known as the "Phos" lamp were, it is true, unsuccessful, and ended in a return to the old and smelly oil-lamps. But the lamp now being employed is a distinct improvement upon the former one, and to all appearances the lamp and light are entirely satisfactory. Fig. 21 shows the appearance of the lamp and its generator. The chief feature of the Lewis generator is the use of paper for keeping the carbide charge separate from the water. A kind of tray is made for the carbide out of thick blotting-paper, and the water is transmitted to the carbide through this medium. The result is a slow and steady evolution of gas, and as one generator with its paper tray will hold 16 to 18 oz. of carbide and will maintain a 25 c.p. burner in operation for nine hours, the usefulness and compactness of the Lewis generator and lamp are apparent. The gas does not, so far as the writer is aware, receive any purification, and therefore it is doubtful whether the Lewis generator and lamp are

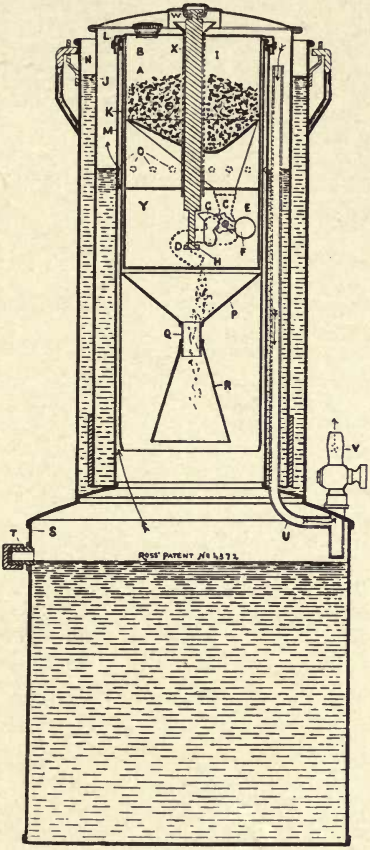


FIG. 22.—The Ross acetylene lamp generator.

suitable for indoor use; but for 'bus illumination purification is perhaps unnecessary.

The Ross acetylene generator and lamp is another form which has been very extensively adopted for motor omnibus and tram illumination.

The question of the purification of acetylene gas intended for use in private houses and public buildings, is, however, of considerable importance. Calcium carbide generally contains calcium phosphide as an impurity, and yields phosphoretted hydrogen on treatment with water. This gas, it is almost needless to say, is very dangerous to health when inhaled, even in its oxidised form of phosphoric acid.

It was formerly supposed that the acetylene gas might be safely burnt as it came from the generator without purification, but it is now recognised that for domestic lighting and for public halls it is absolutely essential that some process of purification should be adopted. The chemicals that have been recommended and tried for this purpose are:—

(*a*) Lime water, by Lundstroem.
(*b*) Bleaching powder, by Lunge and Cedercreuz.
(*c*) Bleaching powder and lead chromate, by Wolff.
(*d*) Acid cuprous chloride solution, by Franks.
(*e*) Chromic acid, by Ullmann.

The first removes all impurities with the exception of the phosphoretted hydrogen, the most dangerous to health; it is therefore useless alone. The use of bleaching powder is condemned by Caro, on account of the possible formation of nitrogen trichloride, and therefore only the methods of Franks and of Ullmann can be considered of practical value. The absorbents adopted in these two methods may be used in conjunction with powdered quartz, and these mixtures have been patented under the names of "Frankolin" and "Heratol" respectively. It is stated that 5½ grammes of chromic acid suffice to purify 1 cubic metre of acetylene gas; but it is evident that the amount required must depend upon the percentage of impurities present. Caro recommends that the purified gas should be subjected to a final scrubbing action with paraffin oil, in order to remove any impurities picked up from the chemical reagents.

The calcium carbide consumed for generating acetylene in the United Kingdom is now estimated to amount to 5,000 tons per annum.

The Calcium Cyanamide Industry.—When calcium carbide is heated in presence of air, the nitrogen of the latter enters into combination with the calcium carbide to form a curious compound called calcium cyanamide, of which the chemical formula is $CaCN_2$. This method of combining, or "fixing" as it is called, the nitrogen of the air was discovered by Frank, of Charlottenburg. It is being exploited by the Società Generále per la Cianamide of Rome, who own the patents covering the process in the greater number of the European countries. The plant required for its manufacture consists of two types of apparatus; namely, the apparatus by means of which pure nitrogen is extracted from the air, and the apparatus by aid of which this nitrogen is fixed in the cyanamide combination. To obtain pure nitrogen an apparatus for making liquid air on the Linde principle is employed. This liquid air is then subjected to fractional distillation in special retorts, for the separation of the oxygen and nitrogen, which have different boiling points. The actual nitrification of the carbide is carried out in retorts filled with the carbide in the form of a powder, and raised to a temperature between a red and a white heat. Under these conditions the carbide absorbs the nitrogen with great rapidity and evolution of heat according to the equation :—

$$CaC_2 + N_2 = CaCN_2 + C.$$

When no more nitrogen is absorbed, the charge is withdrawn from the retort and is allowed to cool in a vessel from which the air is excluded. The product is known in Germany and abroad as *Kalkstickstoff*, or Lime-nitrogen, and is being used chiefly as an artificial manure—a rival of sodium nitrate or ammonium sulphate.

A works for the manufacture of *calcium cyanamide* by this process has been erected at Piano d'Orta, in North Italy, and has now been in successful operation for over 18 months. The capacity in May, 1907, was 4,000 tons per year, and plans for the extension of the works to a capacity of 20,000 tons annually are under discussion.

Similar works are projected or are in actual course of erection at Sebenico, in Dalmatia, at Briançon, near Lake

Geneva; at Martigny, in the Rhone Valley, and at centres of cheap water power in Germany, Norway and America. A total power expenditure of 2 k.w. years is required to obtain one ton of nitrogen in the form of *Kalkstickstoff*. The new product can be applied not only directly to the soil as an artificial manure, but can be utilised for the manufacture of sodium cyanide and of ammonium sulphate. Although calcium carbide is being chiefly employed at present for generating acetylene for illuminating purposes, its application for production of calcium cyanamide is likely to lead to developments of some importance; and in time this application of calcium carbide may overshadow the acetylene gas industry in magnitude and importance.

The use of acetylene gas in the oxy-acetylene blow-pipe for the autogenous welding of metals is another application of considerable industrial interest and value, since temperatures can be obtained with this apparatus which approach those of the electric arc, and the size and shape of the flame are more suited to welding purposes.

This review of the rise and progress of the calcium carbide and acetylene gas industries in the United Kingdom shows that advancement is occurring, and that in certain directions both the calcium carbide and the acetylene gas industries may undergo permanent and successful expansion.

CHAPTER V.

CARBORUNDUM.

I. Historical Introduction.

Carborundum is the trade name given to a carbide of silicium discovered by E. G. Acheson in 1891. The attempt was being made to produce diamonds artificially, by crystallisation of carbon out of its solution in molten aluminium silicate. The experiment was carried out in an iron pot with the aid of a large electric current. It yielded no separation of carbon in the crystalline form, but an extremely hard crystalline blue shining material which was assumed to be a combination of the carbon and alumina (*corundum*), and to which the name of *carborundum* was therefore given.

Analysis of the material showed however, that it had been misnamed, for it was a compound of one atom of silicon and one atom of carbon, and its composition was represented by the chemical formula SiC. Further investigations of the mechanical properties of the compound proved that it possessed great hardness, and its use as a grinding and polishing material was therefore covered by a second patent.

A works for manufacture of the new product was erected at Monongahela Falls. This works commenced operations in 1902, when 1,000 lbs. of carborundum were produced.

The manufacture slowly developed as the properties of the new material became known. A company named the Carborundum Company was formed, and a larger works was erected at Niagara Falls, in 1895. The special feature of this works was the size of the furnaces used for the production of the new material. Up to this date only furnaces of small capacity had been used in the manufacture, but Acheson erected in the new works furnaces for making carborundum which absorbed 1,000 h.p. and thus initiated a very bold step forwards in applied electro-metallurgy. The manufacture of carborundum wheels, and paper, and of grinding tools of special shape was also made a subsidiary part of the Niagara Falls works. The striking growth of the industry is made evident by the following figures from a paper contributed by F. A. J. Fitzgerald to an American Society in 1903:—

Year.	Output in tons.	Remarks.
1891 1892	½	
1893	7·5	
1894	26·0	100 h.p. furnace first used
1895	113·0	1,000 h.p. furnace first used
1896	595·0	
1897	605·0	
1898 . ..	795·0	
1899	867·0	
1900	1,200·0	
1901	1,920·0	
1902	1,870·0	
1903	2,311·0	2,000 h.p. furnace first used

The price of carborundum has fallen during the same period from 40*s*. to 5*d*. per lb. In order to protect the foreign patents the Carborundum Company of Niagara Falls were obliged to erect and operate small factories in France, Germany and Austria. These have been only worked upon a very small scale until recently, and the larger portion of the supply for the European market has been provided by the Niagara works. This method of evading the patent laws and exploiting the European market in the interests of America is, however, now ended, for the German patents granted to Acheson and the Carborundum Company have been declared invalid, and the manufacture of the abrasive has been commenced at Rheinfelden by an independent firm, Messrs. Meyer and Schmidt. The Carborundum Company of Niagara Falls are also erecting a larger works in Germany in an attempt to hold their monopoly, and a further cheapening of the new abrasive may be expected to result from this competition for the European market. The production figures for the Niagara factory for the year 1905 show a decline in consequence of this development of the manufacture in Europe, the figures being 2,518 tons, as compared with 3,177 tons in 1904. Other artificial abrasives produced in the electric furnace have also made their appearance, *Alundum*, obtained by heating bauxite to a high temperature, and *Corundum*, obtained as a by-product in the *Thermit* and *Weldite* processes, being two of these rivals. Carborundum, on account of its greater hardness remains, however, the most efficient of these artificial abrasives.

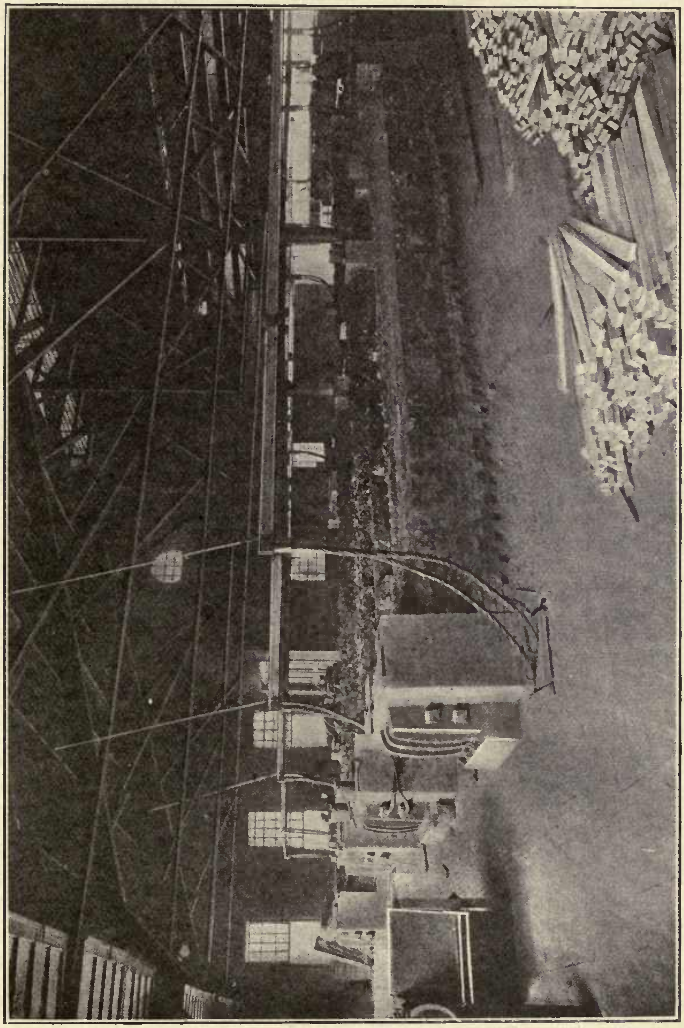


FIG. 23.—Acheson carborundum furnace.

II. Details of the Manufacture.[1]

Carborundum is produced according to the following equation:—

$$SiO_2 + 3C = SiC + 2CO.$$

Silica (or sand) and carbon, yield carborundum and carbon monoxide gas. On account of the great volume of carbonic oxide liberated during the reaction between the carbon and silica, the manufacture is one that demands a well-ventilated building, and considerable care on the part of the workmen.

The sand used in the manufacture is prepared from crushed quartz and contains 99·5 per cent. SiO_2. The purest form of coke is employed as the other raw material, and this is crushed and ground to powder before use. In order to keep the mixture of sand and coke porous when heated, and to allow a free escape for the carbon monoxide gas, sawdust is added.

The 1,000 h.p. furnaces are simply oblong structures 16 feet × 7 feet × 6 feet deep, built up as shown in Fig. 23 of loose bricks, only the ends which carry the current terminals being permanent. The side walls are removed every time a run is completed.

In preparing the furnace for a charge, the side walls are first built up and then the mixture of sand, coke and sawdust is shovelled in, until half the charge is disposed of. A "core" of granulated coke is placed centrally in the furnace, and the remainder of the charge is then shovelled in. The total height of a charged furnace may be 10 feet or more.

---

[1] Taken chiefly from Fitzgerald's paper already referred to.

The furnace is now ready to receive the current, through its permanent end-terminals of large carbon rods in electrical contact with the carbon core. The resistance of this core is at first high, and it will not carry more than 1,800 amperes at 210 volts. As the core and the surrounding mixture becomes heated, however, the resistance falls, and finally becomes steady at about 100 volts. At this pressure the current rises to 7,500 amperes, and it is maintained at this intensity for 24 hours. About half-an-hour after the furnace is started carbon monoxide gas, CO, commences to escape from the top and sides of the furnace, and this gas is ignited in order that it may be burned to the comparatively harmless gas, CO_2. As the furnace temperature rises the evolution of CO increases in volume, until the whole furnace is wrapped in the lambent blue flame produced by burning carbon monoxide.

A 1,000 h.p. furnace charge yields about 8,500 lbs. of carborundum and 6 tons of carbon monoxide gas. About 36 hours are required for the preliminary heating and completion of a charge, and as soon as one charge is finished the current is turned on to the next furnace, which stands charged ready for work. In order to utilise the 3,000 h.p. received from the Niagara Falls Power Company completely in the manufacture of carborundum, the company require three furnaces always at work, and twelve in process of cooling and recharging, that is, a total of fifteen furnaces. The side walls of a furnace are taken down immediately the current is cut off, in order that the cooling may be as rapid as possible.

The furnace is found to contain several layers of material only partially converted into carborundum; these are removed from the crystalline central mass, and are used for charging other furnaces. The crystalline mass in the centre encloses a central core of *graphite*, this being the form into which the high temperature and exclusion of air have converted the coke. After removing all unconverted material, about 4,080 lbs. of crystalline carborundum are obtained per day of 24 hours from each 1,000 h.p. furnace, and the 3,000 h.p. plant, with fifteen furnaces, thus produce about 7 tons of carborundum per day. After removal from the furnace the carborundum is crushed in pan mills, and is then treated with a hot solution of sulphuric acid in lead-lined tanks to extract impurities. It is then washed, and graded into several sizes during the washing and drying operations.

The more recently introduced 2,000 h.p. furnaces are 22 feet long × 8 feet diameter, while the "core" used in them is 36 inches in diameter. The terminals consist of a large number of carbon blocks, each 4 inches square × 36 inches long, separated the one from the other by flat copper strips. The cables conveying the current are connected to these copper strips by a hole into which a heavy thimble holding the end of the cable is bolted. Ten of these connections exist at each end of the furnace, and water is allowed to trickle over the thimbles to prevent fusion.

The current supplied by the Niagara Falls Power Company is at 2,200 volts pressure; this is twice transformed at the carborundum works down to the voltage required for the furnaces.

III. Properties and Uses.

Carborundum is a crystalline material, blue in colour, having a sp. gr. of 3·12 and possessing great hardness. When heated above its dissociation point, the silicon volatilises and the carbon remains in the form of graphite. Its chief use in the arts and industries is as a substitute for emery, and for this purpose it is manufactured in the form of loose powder and grains, wheels, paper and cloth. As an example of its use in the form of grain, granite grinding may be named. For the manufacture of wheels various materials have been experimented with as bond, but a porcelain clay has been found most generally useful. The carborundum grains are mixed with this clay and feldspar in certain proportions, the mixture is moistened with water and then put under a pressure of 2,000 lbs. in strong iron moulds. While still soft the wheels are turned to any particular shape desired, and they are then baked for two or three weeks in special kilns. All wheels are tested by running at a maximum speed before leaving the works. Wheels ranging from $\frac{1}{4}$ inch diameter and intended for dental surgery use, up to wheels 36 inches in diameter, are produced at the Niagara factory of the Carborundum Company; and special grinding tools, such as rubbing blocks, hones, scythe stones, axe stones, knife sharpeners, etc., etc., are also msde.

Carborundum approaches the diamond in hardness, and it is stated to be over three times as effective as emery as a grinding material.

Other applications of carborundum are as a refractory furnace lining, and as a material for introducing silicon into steel.

When used as a furnace lining the carborundum is ground very fine and is mixed in the proportion of three parts by weight of carborundum to one part of sodium silicate. The bricks to be coated with this preparation are brushed clean from all adherent dust, and are then painted with it to a thickness corresponding to the temperature which they are required to stand. They are then left 24 hours to dry and are gradually heated.

According to Acheson, carborundum is infusible under 7,000° F. (3,800° C.), and is unaffected by oxygen, ozone or sulphur under 3,000° F. (1,650° C.).

Tucker and Lampen have published some temperature measurements lately of interest in connection with this subject. The measurements were made with an optical pyrometer. The temperature of the formation of carborundum from silica and carbon was found to lie between

1,920° C. and 1,980° C., while the temperature of its dissociation into graphite was between 2,218° C. and 2,223° C. These temperatures are considerably lower than those originally published by Acheson, and, if correct, they indicate that the field of usefulness for carborundum as a refractory furnace lining is more limited than was formerly supposed.

A German Patent No. 154,536 of 1901 describes the application of carborundum for the manufacture of retorts for zinc distillation, using fireclay as the binding material. The temperature required for this operation is low, and the silicon carbide is unattacked by the zinc fumes. The writer is unaware, however, whether this application has proved successful.

CHAPTER VI.

COPPER.

I. Introduction.

The use of electricity in the copper industry is confined almost entirely to the refining operation; the electrolytic processes for extraction of copper from its ores not having been so successful as their inventors and promoters at one time expected. The electrical processes for the production of tubes and other hollow articles of copper by electrolytic deposition are in reality refining processes, since in these processes slabs of raw copper or granulated copper are employed as anode material. Electro-typing and the production of copper blocks for printing purposes is a special branch of the electrical copper industry, and falls outside the scope of this volume. In principle the method employed for the preparation of "electros" is exactly similar to that used in the copper-refining industry; but special methods of deposition are employed in order to obtain a smooth and fine-grained deposit at the cathode within reasonable time. Electro-typing will not, therefore, be specially dealt with in this chapter, and it must suffice to state here that every large and up-to-date printing office is now equipped with electro-typing plant, and that the time required to make

UNIV. OF CALIFORNIA

blocks fit for use in the printing machines has been reduced from hours to minutes, by the improved methods of working the depositing process.

II. Copper Refining.

Historical Notes.—The use of the electric current for refining copper and other metals was first suggested by Charles Watt in his remarkable Patent No. 13,755 of 1851. A description of Watt's method and cell will be found on p. 48 under "Bullion Refining." No industrial development resulted from the publication of Watt's patent, since at that date the dynamo had not been rendered available for industrial use, and there was consequently no source from which electric currents could be obtained sufficiently large for the conduct of chemical and metallurgical operations.

James Elkington, of the Birmingham firm of electroplaters was the first manufacturer to carry out Watt's proposals for the refining of copper upon an industrial scale of operations.

Elkington patented his method and apparatus in the years 1865 and 1869, and in the latter year he started the first electrolytic copper refinery at Pembrey, in South Wales. The process and plant having worked successfully, the works were enlarged, and it is interesting to note that although the Pembrey refinery has undergone some vicissitudes and has changed hands, it is still in existence, and is still producing copper by the process introduced there by James Elkington thirty-eight years ago.

In the period 1870—1890 the growth of the electrolytic refining industry was slow. Several refineries were erected in England, France, Germany and America, but these were all small in size, and the total output of electrolytic copper in the year 1891 was estimated to amount to only 30,000 tons. Since that year, however, the electrolytic refining industry has undergone enormous expansion, chiefly in America. The process and methods used are similar in all respects to those employed by Elkington in the small refinery at Pembrey in 1869, the only change has been in the magnitude of the operations. At Pembrey the electrolyte was contained in earthenware pots, and the maximum output of refined copper in the early days of the industry was 15 cwts. per day, or 250 tons per annum. To-day, there is one refinery in America which is stated to be producing 200 tons of copper per day, while the aggregate output of all the electrolyte refineries of the world is estimated to amount to 400,000 tons of copper, or to 53 per cent. of the total raw copper production of the world.

This striking growth of the industry has occurred chiefly in recent years, the capacity and output of the American refineries, which contribute over 85 per cent. of the total, having been doubled within the last seven years. The expansion is due partly to the great demand for a very pure form of copper for electrical purposes, and partly to the presence of silver and gold in the American raw copper in sufficient amounts to cover the whole cost of the refining process.

Thirty-six electrolytic refineries are now operating in Europe and America, located as follows:—

America, 11; Germany, 9; United Kingdom, 6; France, 4; Russia, 2; Austria-Hungary, 2; Japan, 2. The largest of these is the Raritan copper works of the United Metals Selling Company, Perth Amboy. This refinery is stated to have a capacity of 120,000 tons copper per annum.

The Refining Process.—When an electric current is passed through a clear solution of copper sulphate containing a little free acid, using plates or thin sheets of pure copper as electrodes, the metal passes into solution at the anode and is deposited at the cathode, while the electrolyte itself shows no change.

If raw copper, as for example that known in the trade as "blister" copper, containing from 96 to 98 per cent. copper, be employed as anode material the same action occurs, but in this case the impurities of the raw copper do not pass into solution, but collect as a precipitate or sludge at the bottom of the cell. At the end of the deposition process, therefore, the copper deposited at the cathode is less in weight than that dissolved at the anode, the difference being due to the insoluble impurities. Only a small e.m.f. of about ·12 volt is required to carry out this transfer of copper from anode to cathode, for the counter e.m.f. due to the deposition of the copper at the cathode is nearly balanced by the direct e.m.f. due to the solution of copper at the anode.

As already stated, the chief changes which have occurred in the electrolytic copper-refining process, since it was first operated at Pembrey, have been in the size and construction of the cells or vats. At Pembrey cylindrical earthenware vessels, 33 inches in height by 17 inches in diameter were employed; each of those pots contained when fully charged four cathodes and six anodes. One hundred of these vessels were placed in series in terrace formation, so that circulation of the electrolyte occurred by aid of syphons and gravity through the whole number, without artificial aid. The electrodes of each pot were connected in parallel, the pots themselves were connected up in series. The output of refined copper from this early arrangement of cells at Pembrey was 15 cwts. per 24 hours or 250 tons per annum.

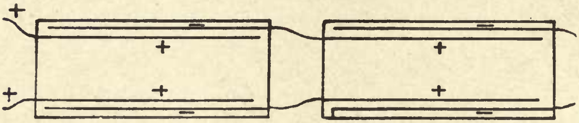


FIG. 24.—The Multiple system of electrode arrangement.

The earthenware vessels soon gave place to large rectangular wooden vats, 4 to 9 feet in length and 2½ feet in width, the number of electrodes was increased from 10 to 40 or 60 per vat, while the weight of each anode plate was increased to 2 cwts. The number of separate vats worked in series was also greatly increased, since it was found to be more economical to use dynamos generating current at a comparatively high e.m.f.

As regards the method of arranging the electrodes in the vats, the parallel system of connecting these up electrically is the plan most generally used, and it is interesting to note that this was the system employed at Pembrey. Fig. 24 shows the plan of this arrangement for two vats only, the electrodes being omitted in order to show the current conductors or "bus-bars" more clearly. The anodes are cast from the raw copper in the form of plates varying from 1½ to 2 inches in thickness, with projecting ears or "lugs" as shown in Fig. 26.

The cathode plates are thin sheets of pure copper, rather larger in superficial area than the anodes. In large refineries a special section of the plant is devoted to the production of these thin sheets, by electro-deposition of copper on specially prepared forms, from which they are stripped when sufficiently thick. The "lugs" of the cathode are made rather longer than those of the anodes, so that they rest upon the outside or negative bus-bar, which is carried along the vats at rather a higher level than the inside or positive bus-bar. The bars are bent as shown—the negative bus-bar of one tank thus becomes the positive bus-bar of the following tank. When the vat is filled with electrodes, arranged alternately and provided with lugs of the required design and length, only ¾ inch of space separates each pair, and every cathode will be found to be in electrical contact with the outer bus-bar, while every anode having shorter "lugs" will be resting upon the inner or positive bus-bar. The parallel system of electrode arrangement described above, is also known as the *multiple* or *Thofern* system.

Another arrangement of the electrodes which has been adopted in America, is that known as the *Series* or *Hayden* system. Fig. 25 is a diagrammatic section of a vat filled with electrodes arranged on this system. Only the two end electrodes are connected to the main current conductors; all the intervening electrodes are known as "secondary" electrodes, and are insulated both from the vat and from each other. The result of this insulation is that the current is forced to travel through the vat by the electrolyte and intervening electrodes, and each of the latter therefore acts on the one side as a cathode and on the other side as an anode. Since it is necessary to strip off the deposited copper from the remains of the anodes when these are withdrawn from the vat, it is necessary to employ anodes with a specially prepared surface on one side for this system of working. The working cost of the "series" or Hayden system is therefore higher than that of the parallel system of working, but notwithstanding this fact, the series system is still in use in two American refineries.

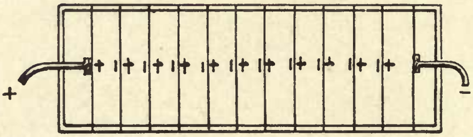


FIG. 25.—The Series system of electrode arrangement.

As regards the important question of current density, this may vary from 13 amperes up to 40 amperes per square foot of cathode area. The higher current density of course lessens the time required for the refining process, but its use is allied with special difficulties due to short circuits, and to deposition of impurities with the copper at the cathode. In the Anaconda (U.S.A.) refinery, a current density of 13 amperes per square foot is employed; this deposits 210 lbs. of copper per 24 hours per vat. A low current density is generally advisable where the anode copper contains much silver and gold, or a high percentage of other impurities, also where the power cost is high; while a high current density is allowable under the opposite conditions.

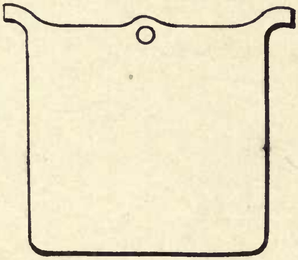


FIG. 26.—Form of anode slab.

The maximum current density which can be employed, without special arrangements for circulating the electrolyte very rapidly, is 40 amperes per square foot.

The maintenance of the copper sulphate solution at its proper strength and purity is the most difficult part of the management of electrolytic refining works. The impurities of the anode copper dissolve and accumulate in the electrolyte, and these, in time, cause disturbances in the process and affect the chemical and physical properties of the copper deposited at the cathode. The electrical resistance of the electrolyte in each vat must be measured, therefore, twice or oftener each day, and tests of the acidity and specific gravity of the copper sulphate solution must be carried out daily by properly qualified men.

The chief progress of recent years in the operation of the electrolytic copper-refining process has been the substitution of machine for hand labour, in all branches of the industry where this is possible. The casting of the raw copper anodes (a most laborious process when performed by hand labour) is now quickly and satisfactorily carried out by casting machines which produce hundreds of electrodes per hour, while the charging and discharging of the vats is quickly effected in most of the refineries with the aid of overhead travelling cranes. The chief improvement on the chemical side of the process has been the addition of a small amount of hydrochloric acid to the electrolyte. According to Carlson this prevents the loss of silver which may otherwise occur.

The impurities of the raw copper which collect on the floor of the vats are known as *slimes*, and are removed from the electrolyte by filtration. The mud obtained from the filters is pressed into cakes and dried, after which it is sent to the bullion works, for extraction of the silver and gold which it contains. It is the presence of these valuable metals in the slimes from the American refineries, that has been the chief cause of the rapid development of the electrolytic refining industry in that country. American copper ores are pyritic in character and carry much larger amounts of silver and gold than the copper ores mined in Europe. Ulke has estimated that 27,000,000 oz. of silver and 346,000 oz. of gold are annually recovered from the *slimes* of the American refineries, and that the value of the recovered *impurities* of the raw copper mined in that country approaches £4,000,000 per annum.

The absence of these valuable constituents in like amount in the copper ores mined in Europe is the chief reason why the European refining industry, though older than the American industry, has shown much less growth and expansion. It is this, rather than any lack of scientific knowledge or ability, that has led to the transfer of the chief centre of the electrolytic refining industry across the Atlantic.

III. Descriptions of Refineries.

Refineries in the United Kingdom.—The following details are drawn partly from the recently published monograph by T. Ulke on "Copper Refining,"[1] and partly from the writer's own notes and information on this subject.

Pembrey, near Swansea, South Wales.—As already stated, this was the first place where copper was refined by the electrolytic process, the works being erected according to the plans of Mr. James Elkington, and put into operation in the year 1869. Full details of the plant and of the procedure adopted in this pioneer works will be found in the two British Patents Nos. 2,838 and 3,120 granted to James Elkington in the years 1865 and 1869 respectively.

---

[1] "Modern Electrolytic Copper Refining." Chapman & Hall, London, 1903.

Elkington employed cylindrical earthenware vessels 33 inches high and 17 inches in diameter, placed terrace-wise, for his electrolytic vats. Each vessel contained four cathodes and six anodes, each anode being faced by two cathodes. The electrolyte—a saturated solution of copper sulphate—was conveyed through the series by gravity, and then pumped back to the highest vessel. One hundred of these vessels were worked in series, and the electrolyte was conveyed from the overflow of one to the bottom of the next by lead pipes. It is noteworthy that in all essentials this arrangement of plant is still followed to-day, and the following extract from Ulke's monograph is of value upon this point:—

"The works established by the father of modern copper refining are in successful operation to-day, due chiefly to the remarkable fact that both Elkington's apparatus and process were well conceived, and needed but little improvement to bring them up to present standards."

The Pembrey works has been much enlarged since these early days of the industry when the output was only 15 cwts. per day of 24 hours, and the earthenware pots have given place to lead-lined vats of much larger cubical capacity.

According to Ulke, the total power now utilised is 385 k.w. The number of vats is 1,065, and anodes 24 inches long × 10 inches wide are employed, with cathodes 24 inches

MINING U. OF C. ASSOCIATION

× 4 inches. The e.m.f. per vat is ·30 volt. The Pembrey works is no longer owned by the Elkington Brothers, but has passed into the possession of Elliots Metal Company of Swansea and Birmingham. The original output of 15 cwts. per day had been increased to 20 tons weekly, or 1,000 tons per annum, when the works were transferred to Elliots Metal Company. According to Ulke, the present output is about 130 tons per week, or 6,500 tons per annum.

Froghall, Staffordshire.—The copper refinery at Froghall is owned by Messrs. T. Bolton & Sons, a very old-established firm of wire-drawers, dating from 1783, and the plant at this place is reported to be the largest in operation in the United Kingdom, having a capacity, when all the vats are in use, of 8,400 tons of copper per annum.

The generating plant at Froghall comprises eight dynamos, each producing 1,500 amperes at 50 volts; the total power available for the vats is therefore 600 k.w.

The depositing vats number 550, and are lead-lined wooden tanks—each measuring 48 inches × 30 inches × 42 inches deep. When fully charged each vat contains ten anodes and nine cathodes. The vats are arranged terrace-wise, and from sixty to seventy vats are worked in series. The anodes are cast from blister copper made at the smelting works carried on by the same firm at Widnes, Lancashire, and are allowed to remain in the vats for six weeks. The refined copper produced at Froghall is remelted and cast into wire-bars, which are then sent across to the Oakamoor wire works for drawing down into wire and tubes, or rolling into plates, etc.

Widnes, Lancashire.—There are two refineries at this place: one owned by Messrs. T. Bolton & Sons, and the other by Messrs. McKechnie Brothers. No very reliable information has been published concerning the latter refinery, but according to T. Ulke the works contains 234 vats, and produces about 9 tons of copper per day.

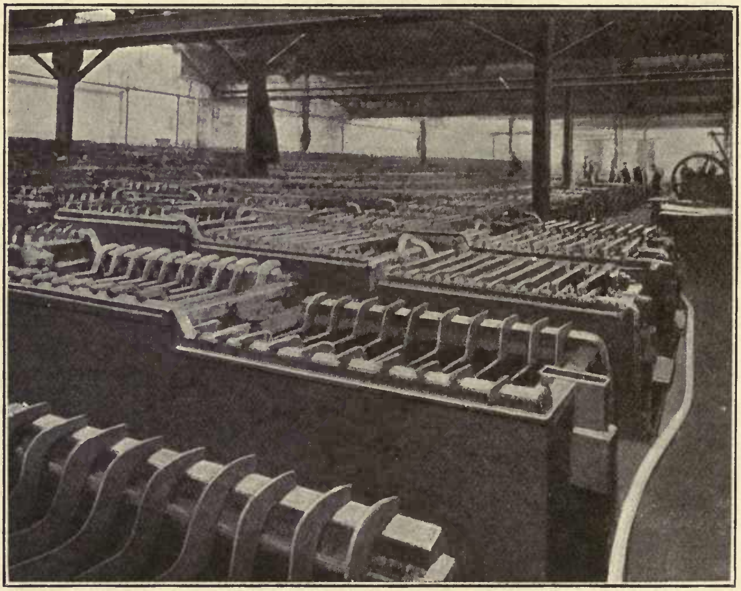


FIG. 27.—The tank house at the Widnes Refinery.

The refinery owned by Messrs. T. Bolton & Sons is a branch department of their smelting works, which is known as the Mersey Copper Works. A large portion of the refined blister copper produced at this works is sent to Froghall for electrolytic treatment, but a portion is retained and refined on the spot. Four dynamos of the Elwell-Parker type supply the continuous current required at this works, these being rope-driven by Galloway engines, each generating 1,500 amperes at 50 volts.

Some details of the vat arrangement will be found in Ulke's monograph. The maximum capacity of this refinery is about 350 tons per month, or 4,200 tons per annum. In March, 1897, the Widnes refinery was stated to be producing 300 tons per month, but since that year no figures bearing on the output have been published.

Swansea.—The small electrolytic refinery at this place forms part of H. H. Vivian & Sons' copper works, and is distinct from the copper, cobalt and nickel works owned by the same firm. This refinery has a capacity from 8 to 10 tons of electrolytic copper per day, and Ulke, in his monograph, gives a few details concerning the casting of anodes, arrangement of electrodes in the vats, and methods of testing the resistance of the electrolyte in use in this works.

According to Ulke, the six refineries located in the United Kingdom contribute 8·8 per cent. of the total world's output of electrolytic copper.

Refineries in the United States.—Newark.—The electrolytic refinery of the *Balbach Smelting and Refining Company* is situated at Newark, and is the oldest in the United States, having been erected early in the eighties. At the end of 1883, the Balbach refinery was producing 2 to 3 tons of copper per day; the present output is over 45 tons of 2,000 lbs. per day, or at the rate of 16,000 tons per year.

The power plant consists of one 300 k.w. and two 125 k.w. machines, the larger generator supplying 4,300 amperes at 70 volts.

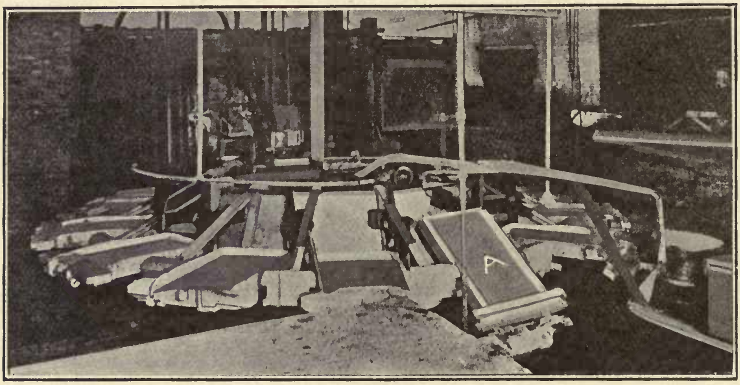


Fig. 28.—Walker and Thum's anode casting machine.

The refinery contains 440 tanks, each containing when fully charged 20 anodes and 20 cathodes, arranged on the multiple system. The anodes are cast 36 inches × 24 inches × $1\frac{1}{4}$ inches thick, and weigh 300 lbs. to 400 lbs. each. The cathodes are 36 inches × 24 inches. The current density used is 16—17 amperes per square foot, and the electrolyte is maintained at a temperature of from 50° to 60° C. Fig. 28 shows the anode casting machine. This is a combination of Walker's standard casting apparatus and Thum's special tilting mould. By means of the latter the partially cooled anode slab is tipped off the mould into water at the point marked A in Fig. 28. The anode drops on to a moving platform immersed in water; this transports it to a carriage which brings it to the refining house ready for use.

Great Falls.—The Boston and Montana Copper and Silver Mining Company run a large electrolytic refinery in connection with their smelting works, at Great Falls, Montana. Power has been obtained from the river by the erection of a dam, which gives a head of 50 feet at the power house. A twin turbine is connected directly to two 810 k.w. generators, giving 9,000 amperes at 180 volts under normal running conditions. The refinery is equipped with 324 vats, each 9¾ feet × 2⅓ feet × 3¾ feet deep. Each vat contains, when fully charged, 20 anodes and 20 cathodes arranged on the multiple system. The anodes are cast 2½ inches to 3 inches in thickness, and weigh each over 600 lbs. A fully charged vat at Great Falls therefore contains 5 to 6 tons of copper; eighteen days are required to work off this charge when using a current density of 40 amperes per square foot. The vats are worked in three sets of 104 each, with an e.m.f. of ·60 volts per vat. The output of copper is about 75 short tons per day, or 27,000 tons per year.

Anaconda.—The Anaconda Mining Company also have an electrolytic refinery in connection with their smelting works at Anaconda, in Montana. This refinery is run by steam power; the generating plant comprises nine dynamos, equal in the aggregate to 2,320 k.w. The number of depositing vats is 1,430, each vat being 8 feet in length × 4 feet wide × 4 feet deep. The vats are constructed of 3 inch planks and are lined with sheet-lead. When fully charged each vat contains 76 anodes and 80 cathodes, hung in a double row. The anodes measure 24 inches × 32 inches × 1¼ inches thick, and weigh each 230 lbs. A fully charged vat contains over 7½ tons of anode copper. Assuming that 1,200 vats are kept constantly in operation at Anaconda, the value of the copper present in the vats is not less than 9,000 × 60=£540,000. This estimate makes no allowance for the value of the silver and gold present in the slimes, but it shows that electrolytic copper refining, as practised in the States, demands an enormous capital outlay for plant and stock, and it also explains the concentration of the industry in comparatively few hands.

The Anaconda plant is worked on the multiple system as regards electrode arrangement, and 200 vats are worked in series. A terminal e.m.f. of 60 volts is employed for each set of vats, representing ·30 volt for each pair of electrodes. A current density of 10 to 20 amperes per square foot of cathode area is used. The daily output of copper lies between 90 and 150 tons, equivalent to an annual production of between 32,400 tons and 54,000 tons.

The anode copper used in these two refineries is practically the same in composition, since it is obtained by smelting the ores of the famous mines of the Butte district of California. This raw copper contains on the average 98 to 99 per cent. copper, with 40 to 120 oz. of silver per ton, and ·10 to 1·50 oz. gold. The refined copper contains 99·96 per cent. Cu, with traces of arsenic, antimony and silver.

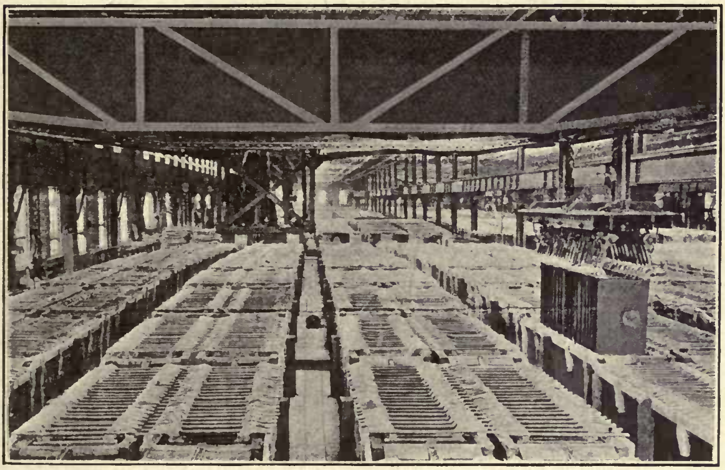


FIG. 29.—The tank house at Raritan Copper Refinery, U.S.A.

Perth Amboy.—The United Metals Selling Company have erected, at Perth Amboy, the largest electrolytic copper refinery in the States—this being known as the Raritan Copper Works. This refinery uses as anode material copper from many of the Western Mining Companies testing 95 to 99 per cent. purity. Copper testing below 99 per cent. is first subjected to a furnace treatment in order to eliminate some of the impurities, and is then cast into anode slabs 24 inches × 36 inches × 1 inch thick. The power required for this works is generated by 5 × 600 k.w. steam-driven sets, each delivering 4,500 amperes at 135 volts with an efficiency of 93·5 per cent. The vat house contains the whole of the electrolytic plant. It measures 600 feet × 200 feet, and contains 1,600 depositing vats, arranged in four groups of 400 each. Four powerful travelling cranes for handling the electrodes run the length of the building; each crane serves 400 vats. The vats are 8 feet in length × 2 feet wide × 2 feet deep. Each contains 22 anodes and 23 cathodes arranged on the multiple system. In order to provide for the circulation of the electrolyte, the vats are also grouped in smaller sets, each provided with a well for the electrolyte and a pump. The thin sheet cathodes of pure copper are made in what is known as the "stripping" vats, of which the Raritan works has 180. The cathodes in these vats are rolled plates of pure copper smeared with grease or plumbago, the edges being protected by grooved wooden strips. The cathodes are kept in the "stripping" vats 36 hours, are then removed, and the thin sheet of deposited copper which has been formed is peeled from the plates. These thin cathode sheets are then flattened out with wooden paddles, and are hung by thin copper loops from the copper rods which rest upon the negative bus-bar of the depositing vat. The anodes when they arrive from the casting house are hung on special iron frames. The travelling crane picks up the frames with its complete set of anodes, and deposits this in the vat prepared for it. The cathode sheets weigh 6 to 8 lbs. when inserted in the vat, and increase to 75 or 80 lbs. during the seven days they are left in the vat. Fresh cathodes are supplied weekly until the end of 42—43 days, when the original anodes will themselves need replacing. The product of the vats, in the form of cathode copper, is melted down and brought to "pitch" in special refining furnaces, before placing on the market. The approximate daily output of the Raritan refinery is from 150 to 200 short tons, equivalent to between 54,000 and 72,000 tons per annum.

Of the remaining electrolytic refineries situated in the United States, that of *Guggenheim Brothers*, at Perth Amboy, and of the *Nichols Chemical Company*, at Laurel Hill, New York, are the largest. The first named is equipped with 816 depositing vats, utilises 1,560 k.w., and produces from 100 to 150 tons of refined copper per day. The second has 120 vats of large size, utilises 450 k.w., and produces 120 tons per day. This refinery employs the Hayden, or series, system.

The most recently erected electrolytic refinery in the United States is located on the shores of Puget Sound, on the Pacific Coast, six miles distant from the city of *Tacoma*. The electric power required here is obtained by long distance transmission from a generating station at Electron, Washington County. The current is transmitted to the works at 40,000 volts pressure, but this is reduced to

100 volts before use. About 200 h.p. are utilised in the various departments of the Tacoma works, and the daily output of refined copper varies from 30 to 43 tons.

The nine older refineries of the United States, according to Ulke, contribute 86·5 per cent. of the world's total output of electrolytic copper.

THE FRENCH AND GERMAN REFINERIES.

Hamburg.—The French and German electrolytic refineries resemble those located in the United Kingdom. They were started in the early days of the industry, and have not developed, as the American refineries, into undertakings of enormous magnitude and extent. The most notable of these refineries is the *Norddeutsche Affinerie* at Hamburg. This works was started in 1876, and is still operating. According to Ulke, there are 600 depositing vats in this refinery, and the most recent estimate of the output of refined copper is 10 tons per day, or 3,600 tons per year.

Eisleben.—The *Mansfeld Copper Company* operate a small electrolytic refinery in connection with their copper smelting works at *Eisleben,* in the Hartz Mountains. This works was started in 1872, and was the first electrolytic refinery established outside the United Kingdom. In 1886 the approximate daily production of refined copper was 2 tons per day, and the present output is stated by Ulke to be only 5 tons per day, or 1,800 tons per year.

The remaining refineries located in Germany and France do not demand any special notice. They are all small, and the percentage contribution of the nine German refineries to the world's output of electrolytic copper is only 2·75 per cent., while the four French refineries are credited with 1·60 per cent.

IV. Costs.

The costs of carrying out the electrolytic process for refining copper have been greatly reduced in recent years by increase in the size of the refining plants, and by the introduction of labour-saving appliances. All the more up-to-date refineries on the other side of the Atlantic are now provided with travelling cranes for charging and removing the electrodes from the vats, and with anode casting machines. There is no doubt that in large refineries most important economies result from the installation and use of these machines. In the early days of the electrolytic refining industry, Peters and Ulke estimated the cost of refining to lie between 33*s.* and 64*s.* per ton of refined copper, no allowance being made in this estimate for the value or cost of working up the slimes containing the silver and gold. It is probable that these costs have been reduced quite 33 per cent. by the introduction of labour-saving appliances, and as raw copper has been accepted for refining in the United States, at a charge of 33*s.* per ton, the margin of profit for the refineries must still be fairly high. There are no trustworthy figures available for publication relating to the corresponding costs in Europe. Since the English and German refineries are mostly small in size, and are operated almost entirely by hand labour, it is doubtful if the refining cost can be as low as the American figure.

As regards the value of the impurities of the raw copper, this varies considerably with the different classes of ores. The copper ores of the Butte district of California are some of the most valuable in this respect, and at Anaconda and Great Falls, according to figures given by Hofman, the value of the recovered silver and gold may rise as high as £10 10*s*. per ton of ore. The average value of the *impurities* of the ores of this district is, however, much below this maximum figure, and may be taken as £6 10*s*. per ton of refined copper. If the refiners were allowed to retain and sell their slimes, or the recovered gold and silver, the electrolytic treatment of copper from the Butte district would therefore be a most profitable operation. It has been usual, however, for the mining or smelting companies to retain their interest in their "impurities," and simply to pay a fixed charge per ton for the refining process, when this is carried out at an independent works.

As already stated, the copper ores mined in Europe are comparatively poor in their contents of silver and gold, and consequently the crude copper does not in all cases pay for electrolytic treatment.

V. Properties and Utilisation.

The enormous development of the electrolytic copper-refining industry in recent years has been partly due to the demand for copper for electrical purposes, and the growth of the electrolytic refining industry has therefore to some extent been an index of the development of the electrical engineering industry.

It is a remarkable fact that although silver possesses a higher electrical conductivity than copper, a few grains of silver distributed throughout the mass of a ton of copper increase the electrical resistance and decrease the conductivity by an appreciable amount. Other metals and elements act even more detrimentally than silver, and therefore copper of the highest degree of purity is demanded for electrical engineering work. The standard of conductivity usually employed in engineering specifications is 100 upon Mathiessen's scale, but owing to the fact that the copper assumed to be pure when Mathiessen carried out his experiments contained small amounts of impurity, it is no unusual thing for copper testing 101, 102 and 103 by Mathiessen's standard to be sold for electrical purposes at the present day.

The following figures show the remarkable expansion of the production and demand for electrolytic copper since it was first placed upon the market by Elkington Brothers in 1869.

Year.	1870.	1891.	1893.	1897.	1903.	1907.
Production in tons.	250	30,000	62,500	160,000	320,000	400,000

VI. THE ELMORE AND OTHER PROCESSES FOR MANUFACTURE OF TUBES AND SIMILAR HOLLOW ARTICLES BY ELECTRO-DEPOSITION.

The copper obtained at the cathode, by operation of the ordinary electrolytic refining process, is coarse and crystalline in structure, and it is necessary to subject it to a metallurgical furnace process before it is placed upon the market.

The Elmore brothers were the first to attempt to combine the refining process with the production of copper at the cathode in a form adapted for commercial or domestic use, and a large number of patents were granted to these two inventors in the period 1888 to 1891 for an improved method of producing tubes and other hollow articles of copper by electro-deposition. The fundamental idea of their process was the use of a revolving shaft or *mandrel* as the cathode, in the electrolytic bath. In order to allow the use of comparatively high current density and rapid deposition of the copper upon this revolving cathode, the Elmores arranged for agate burnishers to travel slowly backwards and forwards over the length of the mandrels, and in this way they obtained smooth and fairly dense deposits of copper upon the latter. At the end of 20 or 30 days, according to the thickness of deposit desired, the mandrel was removed from the electrolyte and by special appliances drawn out of the deposit which had formed around it, a seamless tube of copper of high degree of purity being thus produced.

A company was formed in 1889 for exploitation of the Elmore Patents in the United Kingdom and in Europe upon an industrial scale, and works were erected in quick succession at Hunslet, near Leeds, and by subsidiary companies at Dives, in France, and at Schladern, in Germany. Unfortunately the industrial and commercial development of the Elmore process has been attended by many unforeseen difficulties, both technical and financial, and although the three works named above are still operating, the success of the process has been far below the hopes and expectations of its patentees and of the shareholders in the various companies. The process, however, is an interesting one, and some details are therefore given

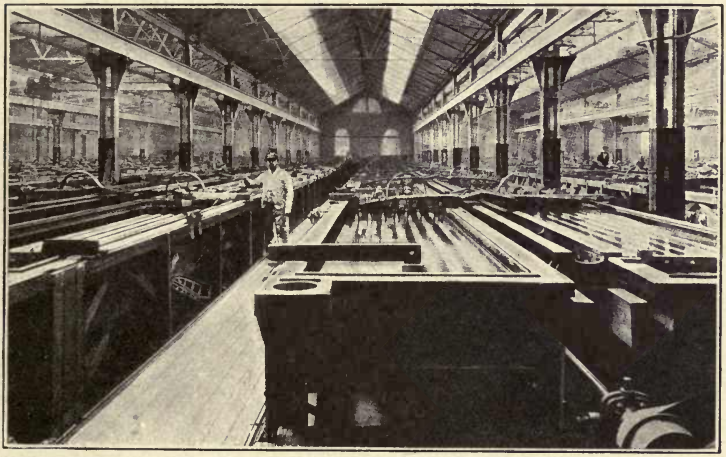


FIG. 30.—The tank house at Leeds Refinery.

of the plant installed at the three works, and of the tests made with the tubes and plates produced.

Leeds.—The generating plant consists of four pairs of Bollinckx engines of 250 h.p., driving by belt transmission Brown-Boveri dynamos, rated at 170 k.w. There are, in addition to these machines, separate dynamos for lighting and power purposes, and the whole generating plant is served with steam from 7 × 300 h.p. Galloway boilers. The depositing vats number 216, and are placed in a special building 265 feet long × 200 feet wide, and containing six bays. The vats are arranged in twenty-four rows of nine each, placed end to end, and when in full work the output of tubes by this plant is said to average 75 tons per week. The rotation of the mandrels and the backwards and forwards motion of the burnishers (one of the main features of the Elmore patents) are secured by electro-motors and link gearing to each tank from an underground main-shaft. The mandrels for tubes up to 4 inches in diameter are of brass, above this diameter the mandrels are of cast-iron, with a brass neck for the current connection. All the mandrels are carefully covered with black lead to facilitate removal of the tube before immersion in the copper sulphate electrolyte. The Hunslet works is adapted for producing tubes up to 4 feet in diameter and 13 feet in length, but the usual size is 12 feet × 2½ inches outside diameter. The latest addition to the works at Leeds is a plant for the production of very thin copper tubes for motor-car radiators. These are made by direct electro-deposition without drawing down from larger sizes. The circulation is kept up by means of pumps, and eight large cisterns are provided, five above and three below the level of the depositing vats. In the latter the slimes are separated and collected for recovery of the gold and silver. The lower cisterns have a capacity of 12,360 cubic feet. The copper used in these works is blister copper containing 96 per cent. Cu, and it is subjected to a preliminary treatment in three refining furnaces, for removal of the arsenic and other impurities. The refined copper is then cast into trough-shaped moulds, 13 feet in length, which are placed in the vats before the mandrel is inserted, and function as anode blocks.

Special appliances are employed for removing the mandrels from the tubes after the deposition is completed, and an overhead travelling crane is used to move about the larger sizes of tube and mandrel. The smaller sizes of tube, in some cases, are produced by drawing down from the larger. This operation improves the toughness of, while it hardens the copper. After drawing down the tubes are heated and then quenched in water.

Besides tubes, the works at Leeds produce calico-drying cylinders, calico-printers' rolls, paper-makers' cylinders, pump-liners, hydraulic ram covers, and other special parts of machinery, in electro-deposited copper.

Schladern.—The works at this place are owned by the *Elmore Metallaktien Gesellschaft,* and they are stated to be producing 1,500 tons of copper tubes per year by a modified form of the original Elmore process. The vats are of wood, are lead-lined, and measure 6 metres in length by 2 metres in breadth. The bottoms are covered with granulated copper to a depth of 200 m.m., and as electrolyte, copper sulphate solution containing 3 per cent. free sulphuric acid is employed. The mandrels run in glass bearings and are placed with their lower side about 30 m.m. above the copper lying in the bottom of the vat. The current strength used averages 200 amperes per square metre, and tubes of all sizes up to 6 metres in length and up to 2·5 meters in diameter are made. The agate burnishing roller moves over the surface of the deposited copper once for every ·03 to ·04 m.m. of increasing thickness. The e.m.f. required is 1·0 to 1·25 volts per vat, and at this pressure 800 amperes pass through the electrolyte contained in each vat. The mandrels are made of an easily fusible alloy, and are covered with tin-foil before use. After the deposition of copper is finished, the mandrel with its coat of deposited copper is placed in a chamber heated by superheated steam, and the alloy is melted out. The deposited copper has a tensile strength of 36,000 lbs. per square inch, and an elongation under strain of 15 per cent. to 16 per cent. The tensile strength is reduced and the elasticity increased by annealing. The highest test result with electro-deposited copper has been obtained with a sample from the Leeds works, which showed a tensile strength of 50,000 lbs. with an elongation of 6 per cent. According to Haber, the Schladern works use a c. density of 250 amperes per square metre for depositing the copper, with an e.m.f. of only ·70 volt.

Dives.—The works at this place is owned by the *Société française d'Électro-métallurgie de Dives*, and is understood to be the most successful of the three Elmore works. The vats are 4 metres in length, while the mandrels have a diameter of from 10 to 40 centimeters. The anodes are cast in the form of triangular bars. The burnishers are hollow cylinders of agate, made in two segments, which automatically lock. These segments embrace the mandrel and its deposit, and travel its entire length in ½ hour. The deposited copper grows at the rate of $\frac{8}{10}$ m.m. per 24 hours, and 1,500 h.p. are required to produce 300 tons of tubes per month. M. Secretan, who founded the works at Dives, has made many improvements in the original process; the details of these, have, however, not been published.

As regards production, the works at Dives are reported to have delivered over 2,000 tons of electro-deposited tubes in one year (1898), but no reliable figures for the present output of this works have come into the writer's possession.

The Dumoulin Process.—To the scientific mind, failures are often more interesting and more worthy of study than successes, and the attempts to make a commercial success of the Dumoulin process at Widnes, in Lancashire, are described here in illustration of this principle.

The Electrical Copper Company was floated in London in

1896, with a capital of £500,000 for purchase of the British and foreign patent rights of the Dumoulin process, for making copper sheets and tubes by electrolytic deposition. The British patents protecting this process were as follows:—

No. 16,360 of 1895; No. 9,289 of 1896; Nos. 2,708—2,712 of 1897.

The process had been worked out experimentally, by M. Dumoulin, at a small works at Brunoy, near Paris, and the Widnes works were planned on a large scale for operation of what was imagined by the promoters to be a successful and economical process.

The writer visited this works in August, 1898, and the following details of the process and plant are based upon the information obtained during that visit.

The generating plant comprised five vertical Peach engines direct coupled to the same number of Crompton dynamos, yielding each 1,300 amperes at 75 volts. The total capacity of the plant was therefore 485 k.w.

The depositing plant consisted of 30 vats, each taking a mandrel 30 feet long × 16 inches in diameter. The mechanical power for keeping these mandrels in slow rotation was supplied by underground shafting with belt drives.

The mandrels were hollow cylinders of copper, capped at each end, and mounted upon insulated bearings. They were only half immersed in the electrolyte of acidified copper sulphate solution, and the chief feature of the

Dumoulin process was the means employed to obtain a dense and smooth deposit of copper during the rotation of the mandrel.

In place of the agate burnishers of the Elmore process, Dumoulin had used strips of sheep-skin held in a fixed wooden frame, running longitudinally over the vat. The strips of skin hung down over the copper deposited on the mandrel, and by a short horizontal movement imparted to the wooden carrier at short intervals of time, every inch of the surface of the slowly revolving cylinder was subjected to their rubbing action.

The precise action of these strips of skin in promoting a dense and coherent deposit of copper upon the revolving cathode was never clearly explained, and when the writer visited the works in 1898 it was admitted that the strips underwent some special chemical treatment before use as burnishers.

The remainder of the plant was somewhat similar to that at Hunslet. The large tubes, however, produced at Widnes were not sold as tubes, but were cut open and flattened out, into sheets 12 feet × 4 feet. Under the usual working conditions, a sheet of copper, 26 gauge in thickness, and measuring 12 feet × 4 feet, could be obtained in this way by electro-deposition in 10 hours, and the weight of the sheet would be 44 lbs., or 14 oz. to the square foot. The current density used was from 35 to 40 amperes per square foot of cathode area. The e.m.f. required was 1·6 volts per vat, but this increased as the anodes became used up.

The sheets of copper obtained in this way were annealed and cleaned by pickling before placing on the market, and the plant at Widnes was designed for producing sixty of these sheets per day. The copper sheets were, to all appearances, dense and coherent. Tests of the copper showed a tensile strength of from 40,000 to 53,000 lbs. per square inch, and an elongation of 28 per cent. to 30 per cent.

The failure of this process at Widnes was due, not to bad quality, but to the fact that the sheets cost more to produce than could be obtained for them on the market, and in July, 1901, the works was finally stopped, and the plant was advertised for sale.

Miscellaneous Processes.—Cowper Coles has patented recently a method of obtaining smooth and coherent deposits of electro-deposited copper by revolving the cathode at a high speed in the electrolyte. The skin friction of the electrolyte upon the surface of the cathode, when this is revolving with a peripheral speed of 1,500 feet per minute, was found to remove all bubbles of gas and irregularities from the surface, and to effect the smoothness and density of the deposit exactly as the agate burnishers in the Elmore process and the strips of skin in the Dumoulin process. A further advantage of the process is stated to be the high current densities which can be employed with it. The process has been operated upon an industrial scale on the continent, tubes 6 feet in length by 1 foot in diameter, and sheets with an area of 20 square feet, having been made by it. The electrolyte is used at a temperature of 50° C. An e.m.f. of ·80 volt is sufficient under these conditions to give a current density of 200 amperes per square foot. Further details of this process will be found in a paper contributed by the inventor to the Faraday Society on July 3rd, 1905.

Krause has described in a recent issue of the *Zeits. f. angew. Chemie* a new method of obtaining smooth and coherent deposits of copper upon a revolving mandrel, which is in use in the works of Langbein & Co., in Leipzig.

The effect produced by the burnishers of the Elmore process, and by the skin friction of the electrolyte in the Cowper Coles process, is in this case obtained by suspending some finely-divided siliceous material in the electrolyte. The grains of this material rub against the surface of the mandrel as it revolves slowly in the electrolyte, and in this way they remove all the gas bubbles and inequalities on the surface of the deposited copper.

Independent tests made with the tubes of electrodeposited copper obtained by this process have shown a tensile strength double that required by the specification for seamless copper tubes drawn up by the German Marine authorities. The best results are obtained with a solution of copper sulphate testing 22° B., to which 5 kgs. H_2SO_4 and 20 kgs. of an infusorial earth are added, per 100 litres of liquid. The mandrel is rotated at a speed of 20 revolutions per minute, and an e.m.f. of 2·5 volts is used to give a current density of 2 amperes per square decimetre. Tubes 2 metres in length by 14 to 20 centimetres in diameter, have been made by this process at Leipzig. The method is protected by German Patents Nos. 175,470, and 125,404.

VII. Electrolytic Extraction Processes.

A very large number of patents relating to the extraction of copper from its ores by aid of electrolysis, have been taken out during the last twenty years. The majority of such extraction processes have depended upon the use of a solvent for dissolving the copper from the crushed and roasted ore, this operation being usually carried out in special *leaching* vats. The solution of copper obtained in this way was then electrolysed in specially designed depositing vats, with insoluble anodes and cathodes of pure copper. Only in a few cases was it proposed to use the copper ore, or matte, directly as anode material in the depositing vats. Most of these processes have failed to attain permanent success, although in some cases large sums of money have been sunk in carrying out industrial trials upon an extended scale. The great difficulty has been due to the fact that the solution obtained, when leaching copper ores, always contained certain impurities, and the gradual accumulation of these impurities in the electrolyte led to irregularities and troubles in the depositing vats.

Only two processes of the above character are, so far as the writer is aware, in operation to-day, and these will be briefly described.

The Hoepfner Extraction Process.—The Hoepfner process is operated in Germany, and depends upon the use of a mixture of cupric and calcium chloride as leaching solution. Ore containing both copper and nickel is usually operated upon, and the copper is, therefore, obtained in the form of *cuprous chloride.* The electrolyte is reported to contain 65 grammes of copper and 80 grammes of nickel per litre, in the form of chlorides ; and before electrolysis it is submitted to a chemical treatment which removes the iron, nickel, lead and silver.

The copper is then deposited by electrolysis in the usual manner, and the solution of cupric chloride which remains is again used for the leaching operation upon a fresh quantity of ore.

The process may also be worked with ores containing only copper ; and with Rio Tinto ore carrying 3 to 4 per cent. copper, good results are said to have been obtained in Germany.

The Hoepfner process has received trial upon an industrial scale at Giessen, at Wildenau, and at Papenberg, in Germany ; but at the present date it is only operated at the latter place.

The Carmichael Extraction Process.—The Carmichael process is an American process which has been worked upon an industrial scale at Dorchester, N.B., Canada, for some months in 1903. In this case a dilute solution of sulphuric acid is used as the leaching agent, and the operation is carried out upon the hot ore as it comes from the roasting furnaces, in large 20-ton vats, lined with lead. The ore used at Dorchester carries only from 2 to 4 per cent. Cu, and this percentage is reduced to ·10 per cent. after the leaching operation.

The solution obtained from the leaching vats contains 2·5 per cent. Cu and much free acid. The depositing vats are provided with anodes and cathodes of lead, and are arranged in cascade fashion. The cathodes are greased and covered with graphite powder to promote the easy removal of the deposit of copper.

The electrolysis of the solution is stopped when the copper contents of the electrolyte have been reduced to 1 per cent.; the solution, which contains all the original free acid, is then again used for leaching fresh ore. The use of sulphurous acid gas in the depositing vats, for diminishing the polarisation of the electrodes and for promoting circulation of the electrolyte, is an important feature of the Carmichael process.

As already stated, the process has been operated in America for some months, and an output of 1 ton of copper per day has been attained. The cost of working the process is stated by the inventor to be about 15*s.* per ton; but it is yet too early to regard the process as beyond the experimental stage of its development. Neither the Hoepfner nor the Carmichael process can be considered to have solved the problem presented by the electrolytic extraction of copper from its ores.

VIII. Electro-Metallurgical Concentration Process.

Recently considerable attention has been directed towards the possibilities of the electric furnace in connection with the smelting of copper ores. During 1903 trials of this description upon an extended scale were carried out at La Praz and other places in France, with Chilian ore, under the direction of M. Vattier, an agent of the Chilian Government. The principle of this method of treatment is to submit the ore, as mined, to the high temperature attained in the electric furnace and to obtain in this way a concentrated "matte," which can then be treated further by the ordinary metallurgical processes of reduction. The process is, therefore, one of electric heat *concentration*, rather than an extraction process. The trials of 1903 in France have shown that it is possible to increase the percentage of

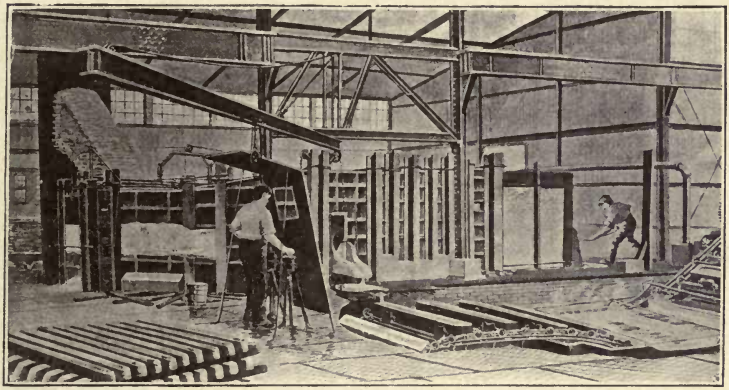


Fig. 31.—Casting machinery for refined copper pigs.

copper from 7 per cent. up to 45 per cent. by this method of concentration without any great loss in the slags which are produced; and it is therefore expected that there will be a great future for the process, in countries where fuel is costly and water power (for generating the electric current) is plentiful. In Chili, where the process is shortly to be tried upon a more extended scale, coke is said to cost £4 per ton, and the cost of smelting copper ore, by the usual metallurgical methods, is reported to amount to nearly £13 per ton of copper produced. The estimated cost in that country, when using electricity for the preliminary concentrating process, is only £3 6*s*. per ton. If these figures are supported by the results of the trials which have since been made there, the success of the electric furnace method of treatment would appear to be assured in Chili, and also in other countries possessing mines of low grade copper ore and abundant water power.

CHAPTER VII.

FERRO-ALLOYS.

I. Introduction.

The discovery of the important effects which small percentages of chromium, nickel, manganese, vanadium or tungsten, either separately or in certain combinations, have upon the physical properties of iron and steel, has led in recent years to the foundation of a new branch of industry, namely that of the manufacture of ferro-alloys. In the early experimental days when the best method of adding one or more of these metals to the molten iron and steel was being worked out, it was imagined that it was necessary to use the purest form of the added metal for this purpose, and attempts were made to obtain pure chromium, nickel, and manganese in the electric furnace for metallurgical use. Apart from the fact that the metals so obtained were contaminated with carbon and silicon, experience has proved that the metals in this form are not well-adapted for putting into the crucible, or pot, of molten iron or steel. Owing to their relatively high melting point, it is difficult to obtain first the fusion and then the diffusion of the added metal throughout the molten mass; and it is now generally recognised that the best results are obtained when the required metal is added in the form of a ferro-alloy. These ferro-alloys, owing to the high temperature required for the reduction of the rare metal, can be produced most satisfactorily in two ways, (1) by use of powdered aluminium and the *Thermit* process of Goldschmidt, and (2) by use of the electric furnace. The first of these methods was referred to in Chapter II., p. 39. It is only necessary to state here, that while the method is still in use at the Goldschmidt works and in Krupp's works at Essen for the manufacture of chromium, manganese and various alloys, it possesses two defects which have prevented its more general use. These defects are, (1) that it is costly, and (2) that all alloys prepared in this way contain a certain amount of aluminium, for it is found to be impossible so to apportion the mixture of oxides and aluminium powder that complete reduction of the former and oxidation of the latter is attained. For these reasons, the electric furnace methods of preparing ferro-alloys are more generally utilised, and in the following pages the more notable works of this kind in Europe and America will be briefly described. The methods in use are based very largely upon those employed by the late famous French chemist Henri Moissan, in his laboratory researches upon electric furnace reactions, and are described fully in his classical book upon this subject.

The manufacture of these ferro-alloys by electric furnace methods is carried on at present chiefly at centres of cheap power in France and Switzerland, the collapse of the calcium carbide industry in 1898—1899 having left many of the carbide works located in these parts unprovided with an outlet for their power. In France, *MM. Keller Leleux & Cie.* are producing ferro-chrome and ferro-silicon in large amounts at Livet and Kerrousse, while the *Société Électro-métallurgique francaise* devotes a portion of their power to the same manufacture at La Praz and St. Michel. The largest works are, however, to be found in the Haute Savoie, on the borders of Switzerland, where the *Société Anonyme Électro-métallurgique Girod* is utilising 18,000 h.p. for the production of ferro-silicon, ferro-chromium, ferro-tungsten and ferro-molybdenum, the aggregate output of the three works owned by this company being given by Dr. Hutton as 9,000 tons per annum, and the value as £360,000.

In Germany *Goldschmidt & Cie.* and *Krupp* are using, as already stated, the aluminium reduction process, in place of the electric furnace at Essen, for producing these ferro-alloys free from carbon.

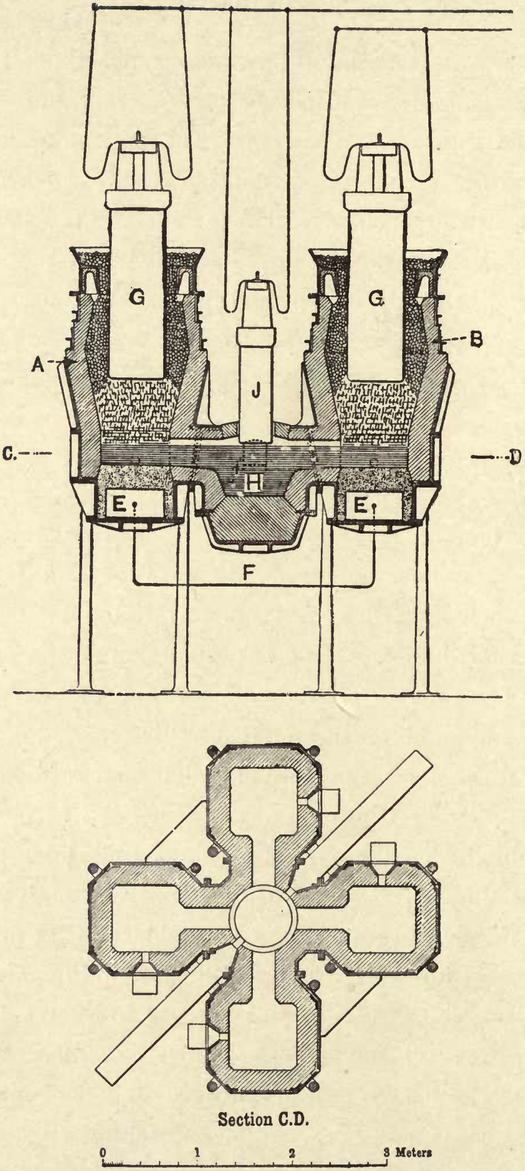
In America, the ferro-alloys industry is less developed; the Willson Company, with works at Kanawha Falls and at Holcombe Rock, Va., are the only producers of ferro-chromium, about 3,000 tons being produced in the two works. Rossi is, however, experimenting at Niagara Falls with electric furnace methods of producing ferro-titanium, and the Titanium Manufacturing Company have bought land for a works at the Falls. A new works has been erected during 1906 at Newmire, Colorado, by the *Vanadium Alloys Company*, of New York, for the manufacture of ferro-vanadium. Recent trials of the ternary and quaternary steels made with the addition of vanadium have proved that these steels are specially suited to the demands made by motor-car work, and it is expected that in time the manufacture of vanadium steel may become a branch industry of considerable importance.

With regard to the use of ferro-alloys generally, ferro-silicon is employed as a de-oxidising agent, while the other alloys are employed for introducing the rarer metal into steel, it having been found that a more homogeneous product is obtained when the rare metal is introduced into the molten iron or steel in the form of an alloy. As an example of this use the manufacture of armour plate may be referred to, all the chrome steel used for armour plates being now made with the aid of ferro-chrome.

The furnaces used for the production of ferro-alloys have been gradually modified in form and details of construction from the calcium carbide furnace installed in the works at the time the new manufacture was commenced. The products obtained in the early days of the manufacture were very impure, and the first ferro-chrome placed upon the market by a French firm contained from 7 to 9 per cent. carbon. Improvements in the design and method of working the furnaces have, however, led to the production of alloys containing a much lower percentage of this element, the percentage of carbon in some of the ferro-chrome now produced having been reduced to under 2 per cent. The use of pure raw materials, the avoidance of excessive heating of the charge, and the prevention of contact between the molten ferro-alloys and the electrodes or the walls of the containing vessel, are the means by which this improvement in the purity of the product has been attained. The use of the electric arc for heating is not conducive to efficient working in most of the processes, since the temperature required lies between that of the ordinary metallurgical furnace and that produced by the electric arc. Furnaces working on the resistance principle have therefore been most successful in the manufacture of ferro-alloys.

II. Descriptions of Works.

Kerrousse and Livet, France.—These two works were founded by Keller, Leleux & Cie in 1900 and 1902, and have between 5,000 and 7,000 h.p. available for the manufacture of ferro-alloys. The larger portion of the power is obtained from the River Romanche, in the vicinity of *Livet, Isere*, by impounding the waters of this river and leading them by an underground tunnel, 1,960 metres in length, to the power house of the works, where a head of 60 metres is thus obtained. Two sets of turbines and generators are installed, the one set generating current at high pressure for transmission to the town of Grenoble, and the other set, generating at low pressure, for utilisation in the neighbouring works. The latter group comprise 5 Negret turbines coupled direct to monophase Thury dynamos, each of 1,200 h.p. capacity. The furnace rooms contain six furnaces each rated at 1,200 h.p., and taking current from



Section C.D.

FIG. 32.—Keller furnace for ferro-alloys.

the low pressure generators, and three furnaces rated at 1,500 k.w.-taking current from the high pressure generators. The maximum daily production when all the furnaces are at work is 45 metric tons of ferro-alloys. The general type of furnace employed is shown in sectional elevation and plan in Fig. 32. The furnace is of the resistance type, and consists of two iron-cased shafts A and B, connecting below by a horizontal canal C. The shafts are lined with a refractory material consisting of a mixture of burnt dolomite and tar, and each is provided with a massive carbon block electrode G, slung on chains, and movable in either a horizontal or vertical direction. The base of each shaft is also provided with a carbon block electrode E, connected electrically by thick copper bars F passing outside the horizontal canal. The vertical electrodes hang for two-thirds of their length within the shaft.

In starting the furnace the charge is introduced between the carbon blocks of the base and the ends of the suspended electrodes. The current passes from one electrode through the charge to the fixed electrode in the base of the furnace, and thence by the external copper conductor to the other shaft, through which it travels in the reverse direction. When the combined arc and electric resistance heating has caused a portion of the charge to be reduced to the molten state, the lateral canal C D fills with the molten ferro-alloy, and the current then passes from one suspended electrode to the other entirely by way of the charges in the vertical shafts A and B, and the molten mass in the canal C D. The suspended electrodes are gradually raised as the charge sinks, and fresh raw materials are added. Any temporary block or stoppage in the lateral canal due to cooling is overcome by use of the secondary electrode shown at J. The vertical electrodes of this furnace are each formed of four carbons of square cross section, the composite electrode built up in this way being 34 inches square by 56 inches in length. The amount of electrode carbon consumed is small, owing to the low e.m.f. used in working the furnaces. In fifteen days two electrodes had only lost 16 inches of their original length; and M. Keller

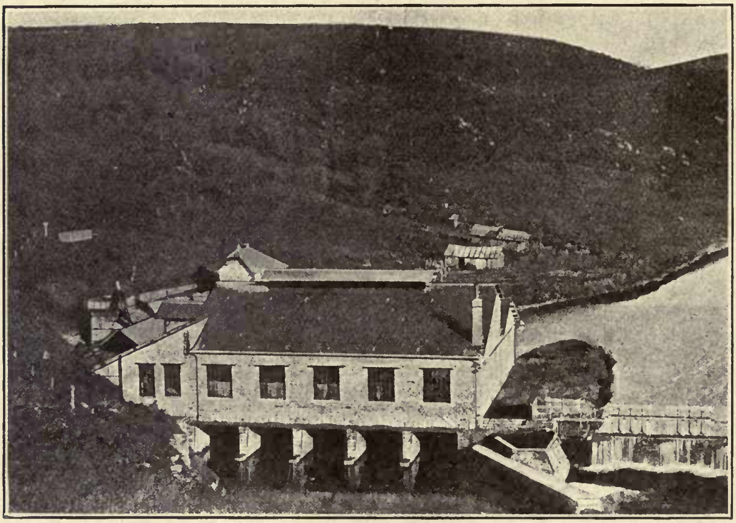


FIG. 33.—The Kerrousse Works, general view.

states that these electrodes can be used twenty days before renewal is necessary.

The Kerrousse works are comparatively small, and utilise a fall of only 8 feet upon the River Blavet, near Hennebont, Morbihan. The generating plant consists of 4 × 150 h.p. turbines direct coupled to 2 × 300 h.p. alternators. The electric resistance furnaces here are smaller than those installed at Livet, and are employed exclusively for making ferro-silicon of high silicon contents.

At the Livet works, *Ferro-silicon*, *Ferro-chrome*, *Ferro-tungsten*, and an alloy containing iron, silicon and manganese, to which the trade name of "*Electro-silico-spiegel*" has been given, are manufactured. Of these products *Ferro-silicon* is made in the largest amount, the percentage of silicon ranging from 25 per cent. up to 75 per cent., and the output being at the rate of 6,000 tons per annum.

Ferro-chrome is produced at the rate of 1,200 tons per year, of qualities ranging from 60 to 65 per cent. of chromium and from 5 to 9 per cent. of carbon.

Ferro-tungsten is only produced in small amounts, testing 75 to 80 per cent. of tungsten and 1 per cent. of carbon.

Electro-silico-spiegel is a product which enters into direct competition with the "*silico-spiegel*" made in the ordinary blast-furnace, and is stated by Keller to be superior to this on account of its much greater purity and lower percentage of carbon. The method of production is patented in Keller's name, and consists in heating ferro-silicon, flint, carbon and the mineral containing the manganese to a moderate temperature in an electric resistance furnace, care being taken that no volatilisation of the manganese occurs. The product contains 22 to 24 per cent. of silicon and 38 to 40 per cent. of manganese, being double the strength of the ordinary "*silico-spiegel.*" It is finding extensive use in English, German and Italian steel works. The works at Livet produce 450 tons of this special alloy per month, or 5,400 tons per year.

The total annual production of ferro-alloys at the Kerrousse and Livet works is thus nearly 14,000 tons.

Ugine, Courtepin and Montbovon.—These three works are situated in the French Alps upon the borders of France and Switzerland, Ugine being in France, while Courtepin and Montbovon are in Switzerland. Between 20,000 h.p. and 40,000 h.p. are available for manufacturing operations in these three works, and the whole of this power is being devoted to the production of ferro-alloys by the Girod furnace and process.

The most important of the works is that located at *Ugine*, although the development of power at this place was only commenced in 1903. The water is obtained from the River Arly, and by means of a dam and conduits 1,840 feet in length, and 52 inches in diameter, a head of 416 feet is obtained at the power house. This contains 18 units, of 8,100 h.p. aggregate capacity. The units are small in size, half being 600 h.p. and half 300 h.p. machines. The turbines are of the Negret-Brenier type, and are direct coupled to the generators, fifteen of which furnish direct current, and three alternating current. The furnace house is 400 feet long, and adjoins the power house. The type of furnace employed is shown in sectional elevation and plan in Fig. 34. The crucible of this furnace A, is made of some refractory material, and is surrounded by silica and graphite in granular form, or of some other material which is of high resistance, and can be employed for heat transfer. Carbon blocks P_1 — P_4 are placed equidistant round this ring of resistance material, and to prevent short circuiting these are separated by bricks of very high electrical resistance.

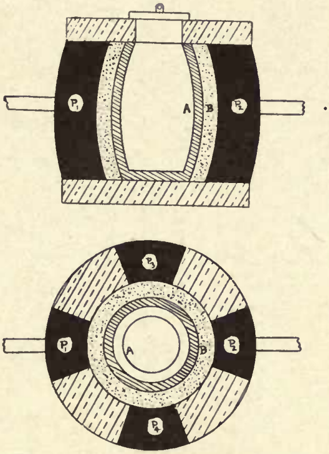


FIG. 34.—Girod Furnace, vertical section and plan.

It has been found that furnaces constructed on this principle show unequal heating, unless means are taken to cause the current of electricity to take the longest path in travelling from the external carbon electrodes to the electrode in the centre of the crucible. In British Patent No. 13,690 of 1905, Girod protects the use of ferro-silicon as resistance material, and the utilisation of different grades of this alloy, in order to obtain a whole series of different resistances. Ferro-silicon can be used in conjunction with graphite, and is available alone up to 1,500° or 1,600° C. The furnaces of this type employed at Ugine are of 200 h.p. capacity, and can produce from 1,000 to 2,000 kgs. ferro-alloys per day. For higher temperatures than 1,600° C. either graphite alone, or the type of furnace shown in Fig. 35, must be employed. The furnace is cylindrical in shape, and is mounted on trunnions for discharging. The furnace body is lined with magnesia brick, while the cover is made of silica brick. The bottom of the furnace is formed of a number of water-cooled iron plates embedded in the brickwork. These act as the lower electrodes of the furnace, and each plate is brought into electrical contact with the contents of the furnace by means of canals filled with molten iron, which run through the brickwork radially from the crucible portion of the furnace. Alternating current is used for this furnace, and the canals are filled with soft iron in the molten state before the charging commences. The

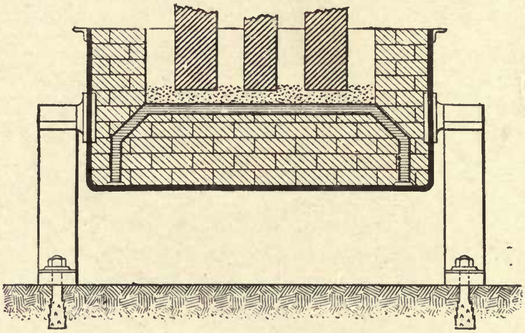


Fig. 35.—Girod Furnace, tilting type.

upper electrodes are four in number, and are suspended with their lower ends just touching the charge of raw materials. These are all connected to one pole of the supply mains, and the sole plates of the furnace are connected to the other.

The works at Albertville (Montbovon) was commenced in 1899 with 1,000 h.p., and 4,500 h.p. was utilised here in 1906, while the third works at *Courtepin* was started in 1903 with 1,800 h.p., and now utilises 5,000 h.p. The total power available for ferro-alloys production in these three works at the end of 1906 was therefore 18,000 h.p., and extensions are in progress which will shortly double this total. As regards the output of ferro-alloys in the three works, the following are the most recent official figures:—

Ferro-silicon . .	50 per cent.	5,000	tons	per annum.
Ferro-silicon . .	30 per cent.	1,000	,,	
Ferro-chromium .	—	2,000	,,	
Ferro-tungsten .	—	800–900	,,	
Ferro-molybdenum .	—	50	,,	
Ferro-vanadium .	—	5 to 10	,,	

The Girod furnace and process yield alloys of specially low carbon contents, and on this account the products of these three works are in steady demand by makers of special steels. The company is the largest producer of ferro-tungsten, two grades being produced, the first with 85 per cent. tungsten and only ·45 per cent. carbon, the second with 72 per cent. tungsten and 2·5 per cent. carbon.

La Praz.—The works of the *Société Électro-métallurgique* *francaise* at La Praz, France, utilises the Heroult furnace and process for the manufacture of aluminium, ferro-alloys and steel.

The works are located in one of the most mountainous parts of Savoy, only 2½ miles from the Italian frontier, and have already been described in Chapter II. (see p. 25).

The Heroult form of furnace will be fully described in Chapter IX., and it is only necessary to state here that the tilting crucible furnace there described has been gradually developed from the simpler form used for ferro-chrome production in the early days of the industry at La Praz. In the manufacture of ferro-chrome, Heroult lines the crucible of the furnace with chrome-ore, and the two heavy carbon electrodes are suspended with their lower ends just dipping into the slag, which covers the molten ferro-alloy. By careful work the amount of carbon present in the finished alloy can be reduced to 2 per cent., and by adding an excess of chromite to the slag, this amount can be still further reduced to 1 per cent. The soft alloy produced in this way is white in colour, and not brittle, although it possesses a crystalline structure. "Hard" ferro-chrome, as usually produced, contains up to 9 per cent. and 10 per cent. of carbon, and is useless for many purposes to which the soft alloy can be applied. An average ferro-chrome contains from 40 to 50 per cent. chromium, and from 2 to 6 per cent. carbon. No figures are available for the output of ferro-chrome at La Praz, but as this was one of the first works to take up the manufacture in France, it has no doubt been an important factor in the development of the industry.

Kanawha Falls.—The Electro-Metallurgical Company of U.S.A. was formed recently in New York to take over and work the Willson patents and furnaces at Kanawha Falls, for production of ferro-alloys. A new works is also to be erected at Niagara Falls, where the manufacture of low carbon ferro-chrome is to be made a speciality. The generating plant at Kanawha Falls comprises three 800 k.w. units, each generating 7,000 amperes at 110 volts pressure. Seven furnaces of the Willson crucible type are installed in the furnace house at this works, and the maximum output amounts to 200 tons of alloys (chiefly ferro-chrome) per month, or 2,400 tons per year. The ferro-chrome produced at this works tests 72 per cent. chromium, 21 per cent. iron, and from 5 to 6 per cent. carbon.

Other companies making ferro-alloys are the following :—Société La Neo-Métallurgie, with works at Giffre; Société des Carbures Métalliques, with works at Nôtre Dame de Briançon ; Bosnische Elektricitäts Gesellschaft, with works at Jaice, Bosnia ; Messrs. Goldschmidt & Cie, with works at Essen ; and the Electric Smelting and Aluminium Company, with works at Lockport, N. Y. ; while at Bitterfeld, in Germany ; at Mattrei, in Austria ; at Sarpsborg, in Norway ; and at Meran, Lonza, and Gampel, in Switzerland, works erected for other purposes have lately been devoting a portion of their power to the production of ferro-alloys.

III. UTILISATION.

Special steels are called *binary*, *ternary*, or *quaternary*, according to the number of elements, other than impurities, which are present. A steel containing only iron and carbon is *binary*, a steel containing iron, carbon, and a third element is *ternary*, while a steel containing iron, carbon, and two other elements is a *quaternary* steel.

It is the *quaternary* steels which are at present attracting most attention. In the following notes it will not be possible to go into this subject at any length, and only the general effect of the ferro-alloys manufactured in the electric furnace, when added to steel, can be dealt with.

The most striking results as regards tensile strength and resistance to shock have been obtained from low carbon quaternary steels containing nickel as third element, and chromium, manganese, tungsten or vanadium as fourth element.

Ferro-chrome is used very extensively in the manufacture of steel for armour plates, tools, and cutlery; it is also employed for high-speed projectiles. According to Hadfield, for ordinary purposes, an alloy comparatively high in carbon may be safely used, but for steels in which a low carbon test is requisite, the higher priced and purer ferro-chrome must be employed. Guillet divides chrome-steels into three classes:—(1) Steels containing up to 7 per cent. of cr.; (2) Steels containing from 7 to 22 per cent. cr.; and (3) Steels containing over 22 per cent. cr. The first class are very hard and resist shock best; class 2 show medium tests both for tensile strength and shock; while class 3 are brittle and of little value.

Ferro-Tungsten.—This alloy is employed in the manufacture of the high-speed tools which can retain their cutting edge, even at a red heat. Messrs. Taylor and White, of the Bethlehem Steel Company, of America, were the first to show what an enormous saving in time and money could be effected in engineering shops by the use of high-speed tools, and the manufacture of the special steels required for these has developed enormously since 1900, the year when the Bethlehem Steel Company's Exhibit at Paris first drew attention to the subject. It is satisfactory to note that Sheffield has attained pre-eminence in this new branch of metallurgy, and that over one half of the ferro-tungsten produced is consumed by Sheffield steel makers.

Ferro-Silicon.—An alloy of iron and silicon, containing 12 per cent. to 20 per cent. of silicon and made by ordinary methods, has been used for many years in iron foundries. The rich alloy containing from 50 per cent. to 80 per cent. of silicon has only been available since the electric furnace replaced the blast furnace method of manufacture in 1899, and its use has undergone great expansion since that year.

The amount of silicon is the chief factor in determining the hardness of cast iron. The high grade ferro-silicon now placed on the market by many firms, therefore, offers an easy and simple means for varying the percentage of silicon in the ladle at will, just before the casting is made. The metal from one cupola furnace can thus be used for a variety of purposes. Silicon also combines with the free oxygen and other gases in the iron, and renders the metal more fluid at the moment of pouring. The silicon in the metal casting finally obtained may range from 1 per cent. up to 2·75 per cent., and according to Outerbridge, additions up to the latter percentage increase the strength of the metal.

Ferro-Vanadium.—This is one of the newer ferro-alloys, upon the value of which metallurgists are not yet agreed. The quaternary steels containing nickel or chromium as third element have, however, shown remarkable changes in physical properties when vanadium is added in only small amounts (·20 per cent. to ·80 per cent.). Nickel-vanadium and chrome-vanadium steels are being used with good results for the axles and crankshafts of motor cars, which have to stand great strains. Tests of over 80 tons per square inch have, in fact, been obtained with the nickel-vanadium steels manufactured in this country.

Ferro-Molybdenum.—This is another alloy upon the value of which metallurgists are not agreed. Molybdenum is said to increase the elongation and elastic limit of steel, and it is therefore being employed on a small scale as an addition to steel designed for all classes of forgings, especially for large crank and propellor shafts. Molybdenum-chromium steel is also being employed with good results for armour-piercing shells, and for high speed tool steel, since it possesses in a high degree hardness and toughness. It is therefore a rival to ferro-tungsten, but it has not yet gained the confidence of steel makers.

CHAPTER VIII.

GLASS AND QUARTZ GLASS.

I. INTRODUCTION.

THE attempts to apply electric methods of heating in the glass manufacture date from 1900, in which year two German engineers named Becker and Völcker, of Cologne, carried out trials with a small experimental furnace heated by electric arcs. The experiments with this furnace were continued by J. Bronn, of Cologne, and led to modifications in its form and to further trials of the improved furnace upon a larger scale of operations at Plettenburg, in Westphalia, and at Mattrei, in the Austrian Tyrol, in the years 1902 and 1903. These trials have not resulted in success so far as the manufacture of ordinary glass is concerned, but a furnace and method for the production of special kinds of glass have been evolved from these early experiments, and vessels of quartz glass are now being manufactured and placed upon the market at reasonable prices, both in this country and in Germany.

Since details of failures in scientific work are often more instructive than those of successes, some space will be devoted to a description of the Becker and Völcker furnaces used in the experimental trials at Cologne, Plettenburg and Mattrei.

II. FURNACES AND METHODS FOR GLASS MANUFACTURE.

The earliest furnace patented by Becker is described in Patent No. 6,944 of 1898. It consists of a small tank furnace divided into three compartments, and of a step hearth upon which the raw materials were heated by three arcs, as they passed down into the first compartment. The furnace used in the trials at Cologne was based on this design, and in its improved form consisted of two step hearths, upon which the fusing of the raw materials occurred by arc-heating, and of a furnace-tank, heated by ordinary fuel. The use of ordinary methods of heating for clearing the glass after the raw materials were fused was illogical and disappointing, but necessary, since this "clearing" of the glass demanded time, and it was obviously impossible to employ electric heating with economy for this stage of the process. By the ordinary method of glass manufacture, in which the raw materials are melted at one end of a large tank furnace, and are then passed under a hanging bridge to the other or working end of the furnace, the glass is exposed for 24 hours to the great heat stored up in the furnace walls and bed, and to the heat obtained from the Siemens regenerative system of burning producer gas. This system depends upon the utilisation of the heat derived from the exit gases to raise the temperature of the incoming gas and air, and it is this utilisation of heat which would otherwise be wasted, that gives the Siemens regenerative gas furnace a very high efficiency as a heating appliance.

The inventors of electric glass furnaces, after a few trials, therefore confined their efforts to attempts to use electricity for the primary melting operation only, and the later furnaces were a combination of electric step-hearths and a gas or coke heated tank. Figs. 36 and 37 show the 70 k.w. furnace of this type, employed by Bronn in the later Cologne trials. The non-success met with in operating this furnace was caused partly by the contamination of the glass with carbon and other impurities from the electrodes, and partly by troubles with the switch gear and measuring instruments, due to the heated atmosphere in which they

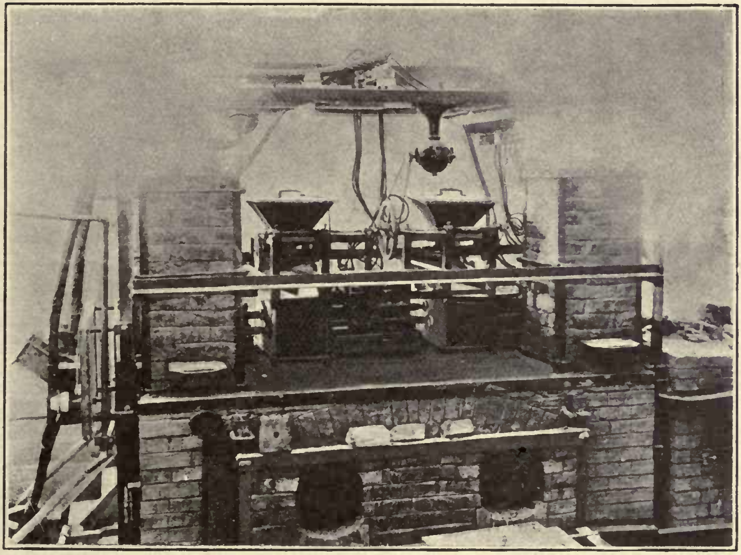


FIG. 36.—Electric glass furnace (front view).

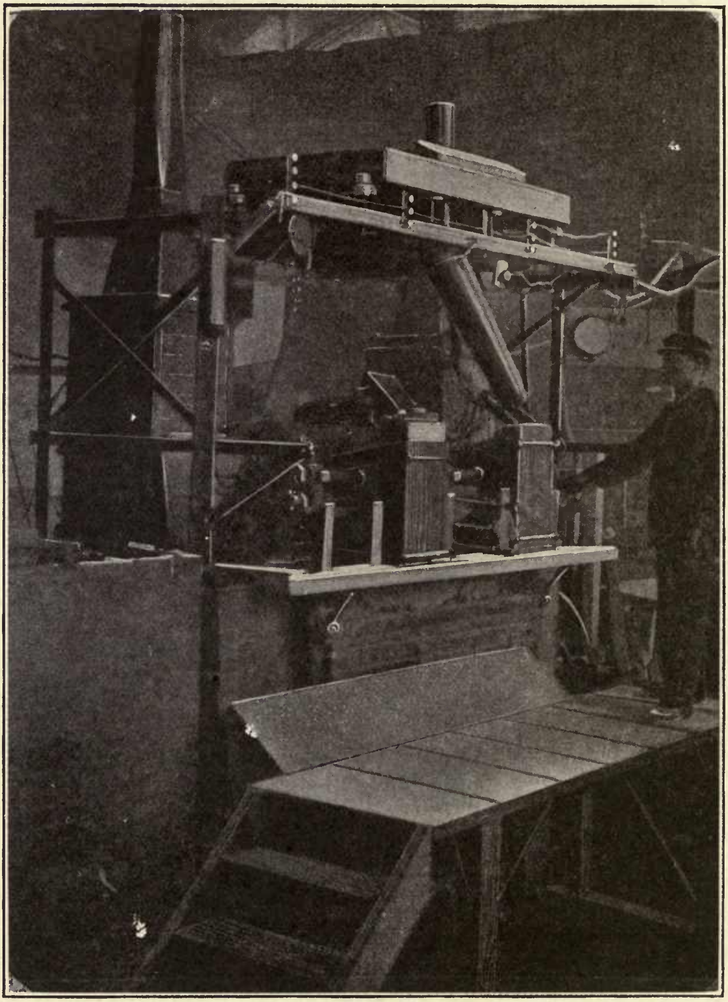


Fig. 37.—Electric glass furnace (back view).

were placed. Various expedients were tried for surmounting the first of these difficulties, namely, the addition of oxidising materials to the raw materials, and the use of brass or iron in place of carbon as electrode material; none of these were successful. Bronn decided finally, from the results of the trials at Plettenburg and at Mattrei, that arc-heating could not be used with any degree of success in glass manufacture, and that resistance heating was more likely to overcome the special difficulties of the problem. This decision was only arrived at, however, after trials of the combined arc and coke heated furnaces at Plettenburg and Mattrei upon an industrial scale. At Plettenburg the trials were made under the joint control of a Brussels Company, *l'Industrie Verriere et ses Derives*, which had purchased all the Becker and Völcker patents, and of a Company which had developed 1,500 h.p. from the River Lenne, and 500 h.p. (steam), at Plettenburg.

The trials at Deutsch Mattrei were under the control of the *Aktien gesellschaft f. Elektro-keramic*, an Austrian Company founded in 1902, with a capital of 400,000 kr. The Company owning the carbide works at Mattrei were interested in this promotion, and placed portion of the 3,000 h.p. developed at Mattrei at the disposal of the new Company, for trials of the various glass furnaces. J. Bronn, in a series of articles contributed to the *Elektro chemische Zeitschrift* in 1904 and 1905, has described the trials at these two centres of cheap power, and has explained the reasons that led to the abandonment of the original type of furnace. As regards the resistance type of furnace, which was evolved as a result of this experimental work, Bronn gives the following particulars :—

The crucible or pot in which the glass was to be heated was surrounded by some suitable resistance material in granular form. After a large number of trials with graphite and other similar materials, that known as *Kryptol* was selected as the best. Using this material and method of heating, it was found possible to maintain the temperature steady at any desired point between the limits of 1,000° C. and 1,800° C., and to heat in this way crucibles and vessels up to 400 cubic centimetres in capacity. When packing the kryptol around the crucible it is necessary to see that the grains are of uniform size, and that they are not more densely packed at one part of the furnace than another. If desired, however, the packing can be designed to give a higher temperature at one point of the crucible than at another.

The e.m.f. for which this method of heating was designed was 100 volts, and as the current density used was high, a special rheostat was necessary to regulate the resistance. This rheostat was made by filling an insulated cylinder with the powdered kryptol, and by using two carbon electrodes, one fixed in the base, and one movable vertically within the cylinder, to give a variable depth of resistance material between the two electrodes. It is self-evident that a furnace of this description cannot be applied with economy to the production of ordinary window or bottle glass. But for special glasses, as, for instance, stained glass for windows, and for other varieties in which contamination with the products of combustion may produce harmful results, the furnace using kryptol as resistance material may prove of considerable utility and value. To what extent the electric resistance furnace just described has been utilised for this kind of work the writer is unable to say.

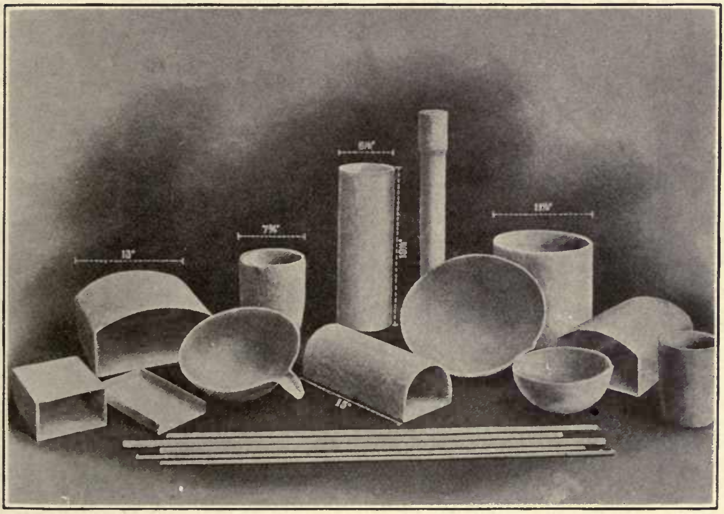


Fig. 38.—Quartz laboratory vessels.

III. Furnaces and Methods for Quartz Glass Manufacture.

Quartz is an impure form of silicic acid (SiO_2), and quartz glass is therefore a glass consisting chiefly of silicic acid, whereas ordinary glass contains silicic acid in combination with lime, soda, potash or lead. The great advantage of quartz glass as compared with ordinary glass is that it has a melting-point 800° C. higher (1,800° to 2,000° C.), and that it is not fractured by sudden changes of temperature. A vessel of quartz glass may be heated far above the melting point of glass (1,200° C.), and may then be plunged into cold water without risk of breaking.

Other important properties of quartz glass are, that it is neither hygroscopic nor soluble in acids, and alkaline solutions affect it less than they affect ordinary glass, except at the higher temperatures.

Glass apparatus and vessels with such remarkable properties are of course likely to prove of great value in laboratory work. The demand for and sale of quartz laboratory vessels is rapidly growing now that these have been placed on the market at comparatively low cost.

The difficulties of manufacture are partly due to the high temperature at which quartz melts, and partly to the bubbles of air which become enclosed in the half-melted mass of material, and only escape after prolonged heating. Messrs. Day and Shepherd, who have investigated this difficulty, find that neither decrease of pressure nor increase of temperature effectually remove the bubbles. The process of manufacture which they recommend is the following:—The mass of quartz fragments is first rapidly raised to a temperature of 2,000° C. in a large bomb type of furnace, the quartz being enclosed in a thin-walled graphite box, which is heated electrically. This temperature causes volatilisation of some portion of the silica, and this silica vapour displaces the air between the grains of quartz. A pressure of 500 lbs. is then applied to the charge by means of an air compressor, and the temperature is reduced to 1,800° C., at which it is maintained for some time. On cooling the graphite box will be found to contain a block of quartz glass almost entirely free from gas bubbles, while any bubbles present are too small to affect the use of the glass for lenses or other optical purposes.

That quartz glass has limitations has been proved by Berthelot, who used for his experiments small quartz tubes made by the aid of an oxy-hydrogen or oxy-acetylene blow-pipe. These tubes were 100 mm. (4 inches) in length by 10 mm. in diameter, and had a wall thickness of 7 mm. Various gases and substances were introduced into these tubes, which were then sealed up, and heated electrically by means of a platinum spiral. Pure nitrogen, oxygen, naphthalene and carbon were heated in this way up to 1,300° C., and it was found that under the conditions of the experiment, the quartz acted like a membrane, capable of endosmose and exosmose. Mylius and Meusser, of the Reichsanstalt in Charlottenburg, have also investigated the properties of quartz glass as regards its resistance to various chemical reagents, and an article dealing with their results will be found in the *Zeitschr. f. Anorg. Chem.*, March 24, 1905.

In Germany quartz laboratory vessels are produced by Heraeus of Hanan, while in England the Thermal Syndicate of Wallsend-on-Tyne have taken up the manufacture. Fig. 38 shows vessels of fused quartz for laboratory use made by the latter firm.

CHAPTER IX.

GRAPHITE.

I. Introduction.

Carbon is found in nature in three allotropic forms—namely, as amorphous carbon, graphite, and diamond. All three forms can now be made artificially, though artificial diamonds are too small to be of any value. The specific gravities of these three forms range from 1·3 to 3·5, graphite occupying the middle position with a sp. gr. of 2·1 to 2·25. Other physical characteristics of graphite are that it leaves a bright metallic streak when rubbed on paper, that it can be machined or cut with a knife like hard wax, and is a good conductor of electricity.

Chemically, graphite is distinguished from amorphous carbon by its behaviour when heated with an oxidising mixture containing chlorate of potash and nitric acid. Amorphous carbon yields under this treatment a brown coloured body soluble in water ; graphite yields an insoluble yellow compound known as graphitic oxide. Graphite is widely distributed in nature, and until a few years ago the purest graphite was obtained from the mines at Borrowdale in Cumberland. These have lately been exhausted as regards the best grades of graphite, and the mines in Ceylon now yield the purest natural product.

It had long been known that amorphous carbon could be converted into graphite by treatment in the electric furnace, but it has remained for E. G. Acheson, the discoverer and patentee of carborundum, or silicon carbide (see Chapter V.), to found a successful industrial process for the manufacture of graphite, upon this method of treatment. When experimenting with the action of high furnace temperatures upon carborundum, Acheson found that it was decomposed into its constituents, silicon and carbon, at a temperature approaching 4,000° C.,[1] and that the carbon separated out in the form of graphite. Further experiments proved that most carbides suffered this decomposition, and that the presence of only a small amount of the carbide-forming base element was required, to convert the whole of the carbon present into graphite. It was the discovery of the latter fact that was the main feature of Acheson's work upon this subject, and enabled his process to become a practicable industrial success. Anthracite coal, containing a fairly high percentage of ash, was

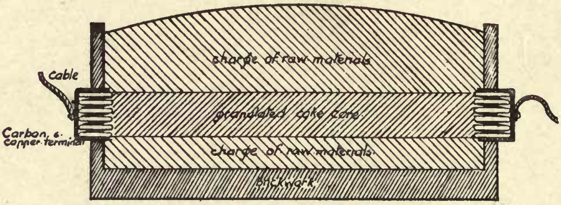


FIG. 39.—Acheson graphite furnace.

---

[1] See p. 99 for correction of this statement.

employed as raw material, and this was simply heated in the resistance type of electric furnace shown in Fig. 39, using a core of carbon rods to carry the current right through the centre of the mass. A current equivalent to 1,000 h.p. was conducted through the furnace and the heating was continued until the silicon and iron carbides formed by the combination of the anthracite with the silicon and iron of the ash were decomposed again, and the silicon was driven off as a gas to the outside of the mass, where it was deposited in the solid form as SiO_2. Graphite then remained in the furnace.

The International Acheson Graphite Company was formed in New York to operate the new process, and a works was erected at Niagara Falls in the years 1897—1898 for manufacture of artificial graphite by the Acheson process. In 1897 the output of artificial graphite by this works amounted to 162,382 lbs., and the totals for the successive years since that date have been as follows:—

1898.	185,647 lbs.		
1899.	405,870 lbs.		
1900.	860,750 lbs.		
1901.	—		
1902.	—		
1903.	2,082,000 lbs.	valued at	$178,670.
1904.	2,932,600 lbs.	,,	$217,790.
1905.	4,591,550 lbs.	,,	$313,980.

II. DETAILS OF PROCESS.

The works commenced with a 1,000 h.p. plant, but in 1904—1905 the plant was increased in size, and a 2,000 h.p. furnace was installed to cope with the rapidly increasing demand for the artificial product. The furnace used for the process takes the form of a long narrow trough, built of fire-brick and lined with a suitable refractory material. At each end of the trough is a terminal, built up of 25 carbon rods, 4 inches square by 34 inches long, to which the supply cables are joined. The trough is filled with anthracite coal, and as this material is a poor conductor of electricity, carbon rods are embedded in the centre of the mass to make electrical contact between the two terminals of the furnace.

The alternating current from the 1,000 or 2,000 h.p. unit is now put on the furnace, and the heating of the charge is continued until the greater portion of the silicon and other impurities contained in the ash of the fuel has been driven off as a vapour. The silicon burns to oxide, and this deposits as a white powder (SiO_2) upon the walls of the furnace. The purity of the graphite depends upon the amount of ash contained in the anthracite used, and also upon the length of time during which the heating is continued. For commercial purposes it is customary to produce a graphite containing up to 10 per cent. of impurities and over 90 per cent. of graphitised carbon, but when necessary the impurities can be eliminated by long continued heating in the furnace, and an almost pure product obtained. The cost of this specially purified graphite is of course much higher than that of the commercial product, since it requires a much larger expenditure of electrical energy for its manufacture. When the heating is finished, the furnace is allowed to cool down gradually before it is opened. The graphite is finally removed and is crushed in a suitable mill; it is then sorted into the sizes required for the various uses to which it is put. The graphite intended for use as a paint is ground to an impalpable powder, while that used for the manufacture of crucibles is crushed to the size of coarse grains or flakes.

As regards the degree of conversion of the amorphous carbon into graphite by the Acheson process, the chemical test described in the earlier part of this chapter has been applied to Acheson graphite, and no amorphous carbon has been discovered; while the resistance to oxidation is far higher in the case of the artificial graphite than in the case of the natural product. The sp. gr. of the Acheson graphite is 2·11.

The chief works of the International Acheson Graphite Company is at Niagara Falls, but small subsidiary works have been erected at La Bathie in France, and at Benetek in Hungary, to protect the foreign patent rights. The most important U.S.A. patent granted to Acheson is No. 568,323 of 1896, and therefore six years have still to elapse before the patents for this interesting process lapse.

Acheson has recently succeeded in making a soft unctuous graphite suitable for lubrication, electrotyping, etc., by heating coal or coke in an electric furnace, with carbide forming material in greater excess than that already present in the ash of the coal or coke. Under these conditions the graphite contained is much softer than that produced by heating anthracite as already described, and it is expected that there will be a large demand for this new artificial graphite. A typical mixture for this process consists of 65 parts ground anthracite and 35 parts of sand and coal, the ash of the latter being calculated as sand.

III. UTILISATION.

The greater portion of the graphite produced at the Niagara Falls works is used for electrical purposes, the Acheson artificial product having been found specially suited for electrodes. During the year ending July 1, 1906, 2,404,171 lbs. of graphite were sold by the International Acheson Graphite Company for electro-chemical or electro-metallurgical purposes, and the consumption in this direction is still growing.

The electrical conductivity of artificial graphite is about four times that of amorphous carbon, and since it can be easily machined and turned in the lathe to any desired form, it has proved a most useful and valuable electrode material. The artificial graphite can be screwed and tapped, or dovetailed like wood, and therefore large electrodes of most complicated form can be built up from smaller pieces or blocks. This facility for tapping and screwing also enables one to use up the small pieces of old electrodes, and to reduce the loss due to old ends of electrodes or "butts." Graphite electrodes have also a much higher resistance to chemical or electrolytic oxidation than the natural graphite, and on that account are specially suited for electrolytic work in which oxygen or oxidising gases are liberated at the anode.

Since round electrodes of small cross section are comparatively expensive if turned down from larger blocks of artificial graphite, a modification of the original Acheson process is employed to produce them. The electrodes are manufactured in the ordinary way from finely ground petroleum coke with pitch as binding material, but certain metallic oxides are added to these materials during the pugging operation. The carbons made from this mixture are then packed in specially constructed furnaces, and are heated to a sufficiently high temperature to bring about the graphitisation of the carbon, the metallic oxides playing an important *rôle* in the changes that occur.

Electrodes which have been made by graphitising carbon electrodes in this manner are, however, sometimes found to be porous, and before these can be used with satisfactory results in certain electrolytes, it is necessary to impregnate the electrode with paraffin. This is best effected by allowing it to stand for some time in a dish containing melted paraffin wax.

Another similar use to which artificial graphite has been put is that of acting as conducting material in the granular mixtures which are now being used for resistance furnace work. The most successful mixture of this kind is *Kryptol*, which contains graphite and the resistance material in various proportions, but in grains of uniform size.

Other uses of artificial graphite are as a pigment, as a substitute for retort carbon in dry batteries, and for small machined bushings and contacts for electrical instruments and machinery. When used as a paint, the finely ground graphite is mixed intimately with linseed oil, and it is important to start with the purest raw materials obtainable. Graphite paint was selected for protecting the cable shields used for the new East River Suspension Bridge in New York, and for this purpose Acheson graphite testing 98 per cent. of carbon was employed. A large number of the graphite paints placed upon the market, however, contain low-grade natural graphites, and the sale of these has prejudiced the more expensive paints made from artificial graphite.

The use of ordinary graphite for crucibles and for stirrers and moulds in metallurgical work is well-known. On account of the ease with which it can be machined and turned, the Acheson artificial graphite can be employed for all these purposes, with advantage to the purity of the metals or alloys finally obtained.

CHAPTER X.

IRON AND STEEL.

I. INTRODUCTION.

THE electric furnace methods of iron and steel production have been developed chiefly by French electro-metallurgists, although the late Sir William Siemens, of London, was the first to patent a crucible for melting iron or steel by aid of the electric current, and to point out the importance of this substitute for crucibles and furnaces heated by coke. The patent (No. 2110) was granted to Sir William Siemens in 1879. Using the form of crucible described in this patent, the inventor was able to melt 10 kgs. of iron in one hour.

Twenty years elapsed before any real progress was made in the introduction of electric heating in the iron or steel industries. The progress, when it did occur, was due chiefly to the failure of another industry. The collapse of the calcium carbide boom in the years 1899—1900 had rendered idle a large number of water-power stations in the south-eastern districts of France. Some new metallurgical application of electric power was therefore urgently required to provide a load for this generating and furnace plant, and to earn interest on the capital expended upon its erection. The production of ferro-alloys was the first step towards the utilisation of this idle plant for other purposes, while in the years 1899—1900 Heroult made his preliminary attempts to produce steel in the electric furnace. Since that year experimental work in this direction has proceeded continuously, a very large number of electric furnaces for iron and steel production having been patented and tried upon a smaller or larger scale of operations in the period 1900—1907. Within the last three years the experiments have entered upon another phase, for the attention of steel-makers in France, Germany and the United States has been attracted to these new methods and processes of steel production, largely owing to the appointment of a Commission in 1904, by the Government of Canada, to investigate and report upon the position of these new methods at that date, in Europe. The Commissioners published their report in August, 1904, this being generally favourable to the new methods. It is now recognised that while there

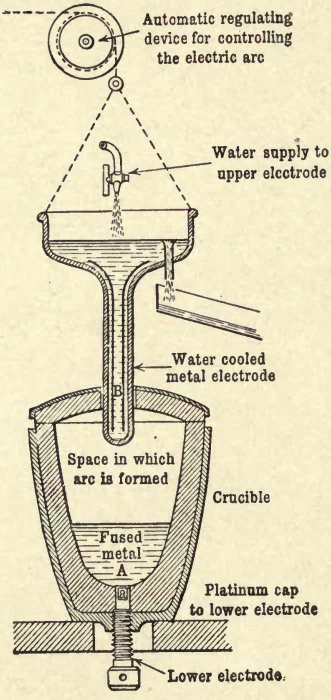

Automatic regulating device for controlling the electric arc
Water supply to upper elcctrode
Water cooled metal electrode
B
Space in which arc is formed
Crucible
Fused metal
A
Platinum cap to lower electrode
Lower electrode.

FIG. 40.—Siemens' crucible furnace.

have been many failures, certain of these new methods have achieved success, and have attained a permanent footing in the iron and steel industry.

The present position (July, 1907) as regards *steel refining* is as follows:—

The *Heroult* process and furnace are in use at La Praz and Froges in France; at Kortfors in Norway; at Remscheid in Germany; and at Syracuse in U.S.A., pig-iron and scrap steel being used as raw materials. The two last-named installations are by independent companies.

The *Keller* process and furnace are in operation at the works of Keller, Leleux et Cie at Livet and Kerrousse, in France, while an independent firm (Messrs. J. Holzer & Co.) are trying a 1,500 h.p. Keller steel refining furnace at their large steel works at Unieux, Loire.

The *Gin Furnace* is undergoing trial at Plettenburg, in Westphalia.

The *Stassano* process and furnace are in operation at the Italian Royal Arsenal at Turin, while a separate company interested in the Stassano patents is carrying on independent experimental work with the furnace in Turin.

The *Kjellin* process and furnace have been operated with success at Gysinge, in Sweden, since 1901, and installations of this type of furnace have recently been made at Krupp's works at Essen; at Gurtnellen, in Switzerland; at Vicar's works, in Sheffield, in England, and also in the United States.

The *Colby* induction furnace is being worked near Philadelphia, U.S.A.

As regards *iron-smelting*, the progress made by the new methods is less striking; but the Keller furnace and the Heroult furnace have achieved satisfactory results during trial runs in France and Canada, made in the presence of independent authorities.

Many thousands of tons of steel and several hundreds of tons of iron have been produced by the more successful of these new electric furnace processes since the experimental trials were initiated in 1899; and at the present moment there is great activity in this new branch of electro-metallurgy.

In the following pages brief descriptions of the more important furnaces and processes will be given in chronological order.

II. Descriptions of Furnaces and Processes.

The Stassano Furnace and Process.—The first patent for an electrically heated furnace for producing steel from iron ore was granted to Captain Stassano, an Italian Artillery officer, in 1898, and experimental trials were commenced with the furnace at Rome in 1899.

These trials being attended with success, a company was formed in Rome for financing a continuance of the trials upon a larger scale of operations, and a plant for utilisation of 1,500 h.p. was planned and partially erected at Darfo, in Northern Italy. This plant was visited by Dr.

Goldschmidt on behalf of the German Patent Office, and seen in operation by this authority in 1901. Unfortunately the company financing the industrial trials at Darfo came to an end of its resources before the process and furnace had entered upon the profit-earning stage of its development, and the trials at this place were suspended in 1902. The Italian Government, however, were interested in the process, and arranged to carry on further experiments with the Stassano furnace at the Royal Arsenal, Turin. The plant erected here has been used for the production of the special mild steel used for shells almost continuously since the year 1903. A new company, *The Forni Termo-elettrici Stassano*, has also been formed to take over the assets of the earlier company, and a plant of four furnaces for utilisation of 2,300 h.p. is now partially erected at Turin. The plant when completed will comprise two 1,000 h.p. furnaces, and one each of 200 h.p. and 100 h.p.

The Stassano furnace has undergone many modifications since the early trials at Rome, and in this description only the two latest forms of the furnace can be dealt with. *The Rotary* form of furnace is shown in sectional elevation in Fig. 41. The furnace is of the arc type, and consists of an outer iron cylindrical casing, lined with magnesite brick and covered with a dome-shaped roof. The heat is obtained by radiation from the electric arcs formed in the centre of the furnace, between the three carbon electrodes. The carbon-holders are water-jacketted, and the movement of the

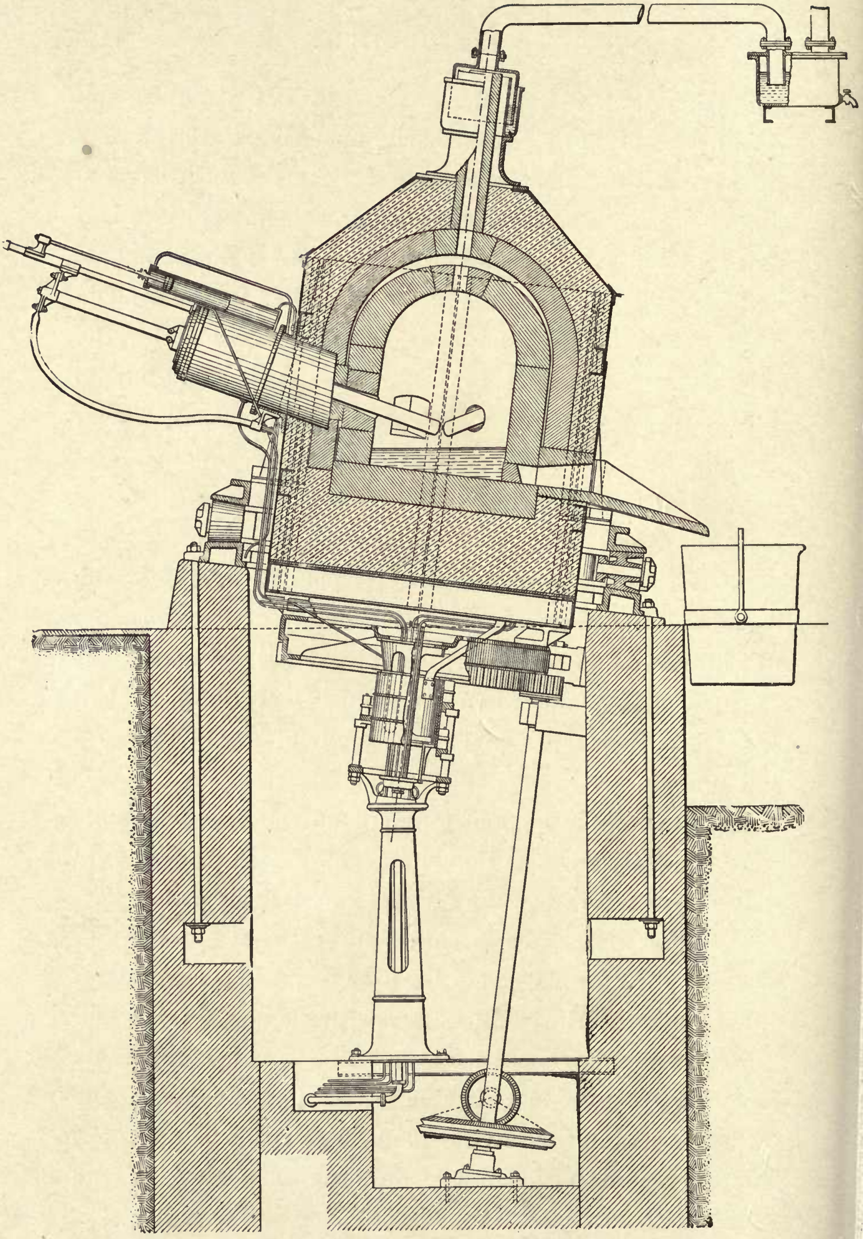


FIG. 41.—Stassano rotary furnace.

electrodes is controlled from a distance by water pressure. The furnace is mounted upon rollers with an inclination of 7° from the vertical, and can be rotated about its axis by means of the gearing shown in Fig. 41. The raw materials are charged into the furnace by the hopper, the arcs are set in operation, and the furnace is then rotated to cause thorough mixture of the molten materials upon the hearth. A three-phase current of 400 amperes at 90 volts is employed to work the furnace of this type at the Royal Arsenal, Turin, and the charge consists of iron and steel scrap in calculated proportions.

The furnace produces 2,500 kgs. of steel in four heats per twenty-four hours, and consumes 850 k.w. hours per charge of 625 kgs. The consumption of electric energy is therefore 1·36 k.w. hours per kg. of mild steel.

A larger furnace of the same type is being constructed for the Forni Termoellettrici Stassano Company, and of this furnace Major Stassano has published the following particulars:—

Cost . .	25,000 fr.
Output . .	4 to 5 tons per day.
Current .	4,900 amps. at 150 volts.
Electrodes .	Four, each 1·5 metres in length by 15 centimetres in diameter.
Arcs . .	Two arcs, each taking 2,450 amps.
Lining . .	Magnesite brick.

It is expected that this furnace will show marked economy in electric power as compared with the earlier types, since

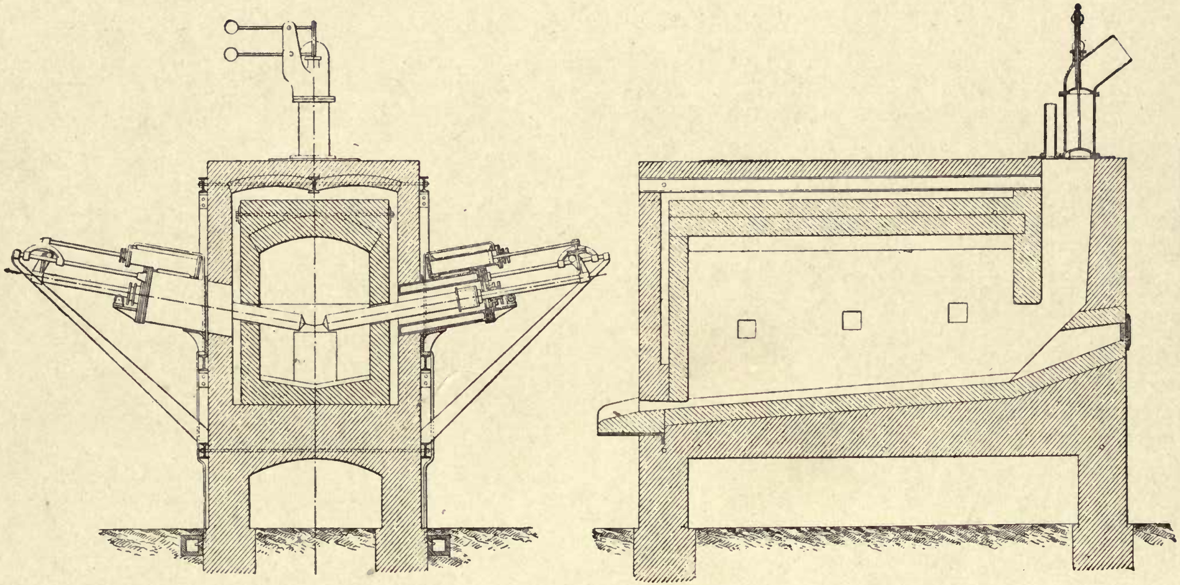


FIG. 42.—Stassano fixed type of furnace.

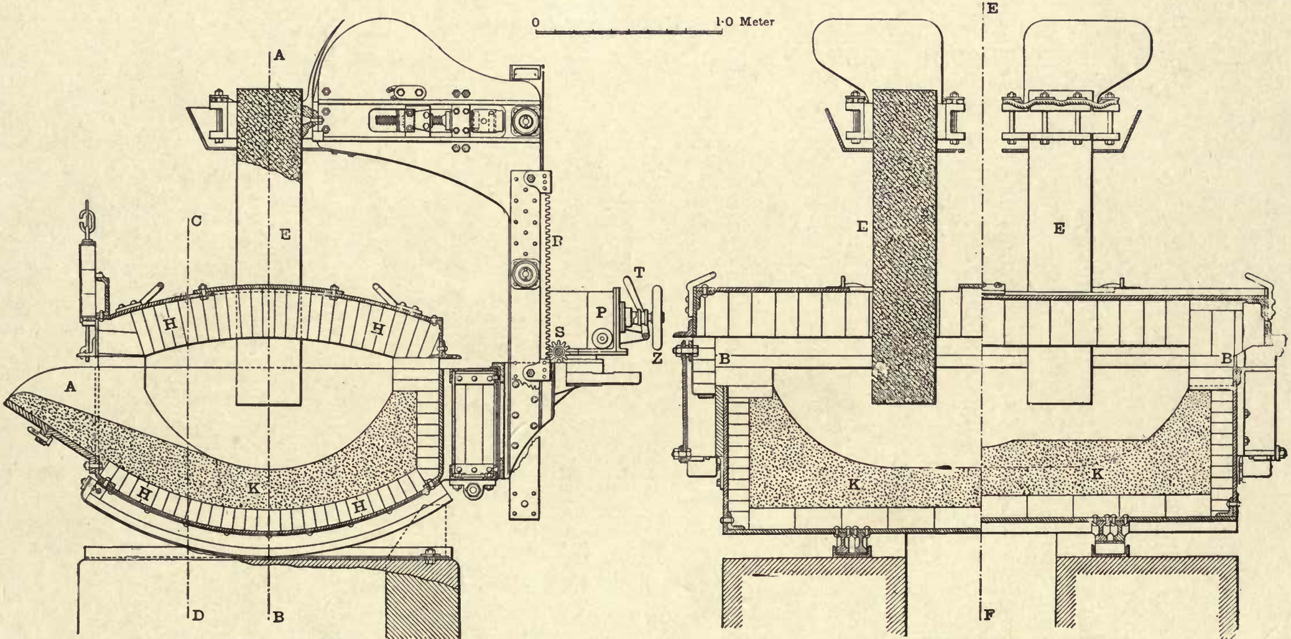
one arc only can be used for certain stages of the melting and refining processes.

In order to still further reduce the energy consumption per ton of finished steel, Stassano has designed the furnace shown in section in Fig. 42. This furnace is known as the "*fixed*" type, and resembles in some respects the ordinary Martin steel refining furnace, radiated heat from arcs between two or three pairs of carbon electrodes being employed to melt the charge upon the hearth. No details of the trial results with this furnace have yet been published.

It must be noted that while in the earlier trials of his rotary furnace at Rome and Darfo Stassano had in view the production of finished steel in one operation from iron ore, the later trials seem to have been confined to the production of steel from pig-iron and scrap. The cost of grinding and briquetting the ore, lime and coke charged into the furnace is believed by the author to have proved too costly for continuance of the attempts to apply the furnace directly to the smelting of iron ores.

The Heroult Furnace and Process.—M. Paul Heroult is the French electro-metallurgist, whose name is intimately connected with the development of the aluminium industry, since his method of obtaining this metal by electrolysis of the oxide was first patented in 1887, and is now generally employed by all the aluminium works in Europe (see Chapter II.).

In the year 1899 Heroult turned his attention to the



Transverse Section E.F

Longitudinal Sections AB. & CD.

FIG. 43.—Heroult crucible furnace.

production of ferro-alloys in a modified form of the furnace he had used at La Praz for calcium carbide production, and it was the success of this application that suggested to him the use of the same type of furnace for steel refining.

The Heroult steel refining furnace is shown in sectional elevation in Fig. 43. It consists of a shallow iron tank lined with dolomite brick, excepting at the openings, where magnesite brick is used. The hearth of the furnace is formed of crushed dolomite, rammed down on top of the dolomite brick lining. The furnace is provided with a curved bottom and is mounted on cogged bars, so that it can be tipped sideways and held at any desired angle for discharging purposes. At the opposite side from the discharge opening there is an inlet for the air blast, and also an insulated supporting framework for the two massive carbons which form the electrodes. These are each 14 inches square by 67 inches in length, and they are capable of being moved either horizontally or vertically by the worm gearing shown in Fig. 43. The top of the furnace contains openings for charging the raw materials, for the two electrodes, and for the escape of the waste gases.

The charge is heated by the formation of arcs between the two electrodes and the surface of the materials on the hearth of the furnace, and also by the resistance of the molten slag and metal. When desired the formation of arcs can be altogether suppressed by lowering the electrodes until they dip beneath the surface of the slag; the heating is then entirely due to the resistance of the molten materials to the passage of the large current (4,000 amperes) employed.

The method of using this furnace for steel production from pig-iron and scrap is as follows:—A mixed charge of pig and scrap in suitable proportions is placed in the furnace, iron ore and lime being added in calculated amount to produce a slag. This charge is now raised to the melting point by combined arc and resistance heating; when this stage is completed the slag will cover the surface of the metal and protect it from atmospheric oxidation. The air blast is then turned on, and the temperature is

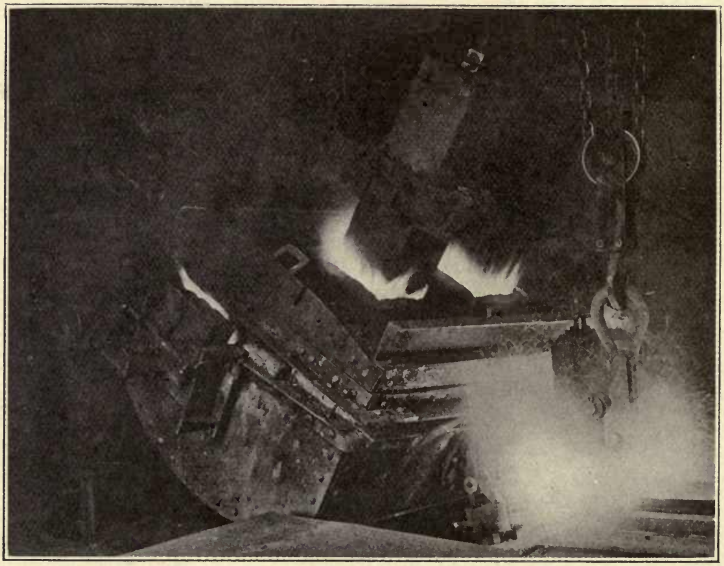


FIG. 44.—Heroult crucible furnace (discharging).

increased to the maximum possible, in order to oxidise all the impurities of the charge, and to drive them into the slag. The slag is poured off and renewed once, twice or thrice during the refining operation, fresh slagging materials being added as required. When the metal has been sufficiently purified by this method of oxidising the silicon, phosphorus and sulphur, and removing these oxides with the slag, a calculated amount of carburite (iron high in carbon) is added, and the finished steel is tipped into the casting ladle.

The ordinary crucible furnace of this type used at La Praz produces 6 tons of steel per day in 2 heats of 3 tons each, a current of 4,000 amperes at 120 volts being employed. The finished product contains on the average ·003 per cent. phosphorus and ·007 per cent. sulphur, while the carbon and silicon contents can be varied at will by regulating the amount and composition of the alloy, added towards the end of the refining operation. The electric power required to produce 1 ton of steel in this furnace is ·153 e.h.p. per short ton (2,000 lbs.). Heroult has stated recently, that by the use of water-cooled electrodes, a considerable reduction in the running costs of the furnace has been effected; and that steel containing only ·01 per cent. of sulphur and phosphorus can be produced from scrap containing comparatively large amounts of these two elements. The Heroult steel refining plant erected by the Halcomb Steel Company, of Syracuse, U.S.A., is designed for the production of 80 tons of steel per day, the raw metal being superoxidised steel from a Wellmann open hearth furnace.

The Heroult Iron-Smelting Furnace used for experimental work at La Praz, and at Sault Sainte Marie, in Canada, is of very simple construction, being merely an iron column of square cross section with a fixed electrode in the base, and a movable one slung vertically in the centre of the column. The box or column is lined with refractory material. The electrodes are of carbon, that fixed in the base of the column being a flat plate, while the movable one is a block of half the sectional area of the column, and 6 feet in length. The method of operating the furnace is as follows:—

The furnace, if cold, is first heated by allowing the arc to form between the two electrodes, and to radiate its heat for 16 to 20 hours. The charge of raw materials, consisting of finely divided iron ore, lime or other fluxes, and coke is well mixed and is placed in the lower part of the column. The upper electrode is then lowered until it touches the charge, and the current is switched on to the furnace. By regulating the distance between the electrodes, the current is kept as nearly as possible constant at 5,000 amperes. As the charge melts and is reduced fresh raw material is added from above. When a sufficient weight of iron has accumulated upon the hearth of the furnace, the tap hole is opened, and the molten iron is allowed to run into the moulds placed to receive it. Attempts to use the hot gases, for preliminary heating of the raw materials, have not proved successful, and for the present this addition to the process has been abandoned.

The trials with this process and furnace for iron-smelting, carried out at Sault Sainte Marie, in Canada, early in 1906, were under the control of M. Heroult himself. About 55 tons of pig-iron were produced in the trials, from various ores of the district, the first runs being made with ordinary Hematite ore, and the Canadian ores, high in sulphur and nickel, being employed for the later experiments. Wood charcoal was also used with success as a substitute for coke in many of the trials. The results of these trials of the Heroult iron-smelting furnace in Canada are considered by those interested in the experiments to have been most promising. Heroult estimates that good pig-iron can be produced there by his process, at a cost of £2 per ton. A permanent installation of the furnace at Sault Sainte Marie for smelting the nickel-iron ores of the district is therefore now under consideration, and if sufficient capital can be raised, a large electro-metallurgical smelting works will shortly be established on the narrow neck of land separating Lake Superior and Lake Huron. It may be explained that a large water-power scheme has been carried out at this spot, and that between 30,000 and 50,000 h.p. have been rendered available for industrial purposes. With cheap water-power, and iron ores at 5*s.* per ton, the new method of iron-smelting starts, therefore, under very favourable conditions, and it will be interesting to see whether, with so much in its favour, it is able to command ultimate success.

The Keller Furnace and Process.—The Keller furnace and process for iron and steel production have been patented and developed by the French firm of *Keller, Leleux et Cie,* this firm being one of the first in France to take up the manufacture of calcium carbide. This firm had been for many years engaged in the production of calcium carbide and ferro-alloys, when in the year 1900 they turned their attention to the application of electric methods of heating in the iron and steel industry. The experimental work has been carried on chiefly at Livet, where 15,000 h.p. have been rendered available by extensive hydraulic engineering works (see Chapter VII. on ferro-alloys). It was here that the Canadian Commissioners witnessed trials with the Keller furnace and process in March, 1904, the other works of the company at Kerrousse, Morbihan, being devoted to the production of ferro-silicon. The principal trials at Livet have been those directed towards the smelting of iron-ores, with the type of furnace already described in detail in Chapter VII. (p. 145).

This furnace is of the two-shaft type, and it can be employed for combined arc and resistance heating, or for the latter alone. Two sets of experiments were carried out by the Canadian Commissioners at Livet with the 1,000 h.p. furnace, grey, white and mottled pig being produced. The furnace was operated as follows :—The ore, coke and limestone were roughly broken in a crusher having its jaws fixed at 2 inches, and after mixture the charge of 12,000 to 15,000 kgs. was fed into both furnace shafts, until these were two-thirds filled. The electrodes were then lowered down on to the surface of the charge, the current was switched on, and the first operation of melting the charge commenced. The current passed down one shaft and up the other, by aid of the small pieces of ore and coke, and by numberless small sparks where any air gaps intervened. The passage of the current from one base electrode to the other was by aid of the external copper conductor. As the heat rose, and the iron was reduced to the molten state, the collection of molten iron in the lateral canal, joining the two shafts, reduced the resistance, and the e.m.f. between the two terminals of the furnace fell, while the current increased.

The variation was however not great, and the electrical switches of several furnaces could be controlled by one man. Since the external copper conductor was always ready to carry the current from one shaft to the other, the furnace might be worked continuously, and all the molten iron might be drawn off from the hearths and lateral canal at short intervals of time. The Keller furnace is operated with alternating current, and in the trials made at Livet, the Canadian Commissioners produced 16 short tons of pig-iron in two runs, with a mean power expenditure of ·350 e.h.p. year, and a total cost (exclusive of the royalty—to the patentees) of £2 8*s*. 3*d*. per ton of 2,000 lbs. It is expected that further economies in the power consumption and cost of labour will be obtained with a larger 2,000 h.p. furnace with four shafts and one central canal, which has been constructed at Livet since the date of the Canadian Commissioners' visit. This furnace is designed for the production of 20 tons of pig-iron per 24 hours, but no figures have yet been published to show whether this output has been attained.

As regards the composition of the pig-iron obtained from the Keller furnace at Livet, the percentage of sulphur and phosphorus was low, while the percentage of silicon and manganese was high. Keller claims, however, that his furnace gives him complete control of the amount of carbon and silicon left in the finished charge, and that as regards other impurities, these will not be higher than the percentage found in the pig-iron from blast-furnaces, using the same raw materials.

Although many hundreds of tons of pig-iron have been produced at Livet by the Keller process, no installation of the furnace is known to the author outside the Livet works, and the production of pig-iron from the ore, by the Keller furnace, must therefore still be regarded as in its experimental stage of development. M. Keller has stated that it will first receive trial upon an industrial scale, in countries which possess iron ores, but no coal with which to smelt them, and it is expected that the first practical trial of the Keller furnace and process will be in South America or in New Zealand.

As regards *steel refining*, Keller has designed a furnace which largely resembles that first employed by Heroult, and it is well known that the latter considers the Keller steel refining furnace an infringement of his own patent. The Keller furnace consists of a fixed chamber lined with some refractory basic material, and heated by two massive carbon electrodes suspended vertically above the metal in the furnace. These electrodes are just allowed to touch the surface of the molten slag, but not to dip beneath the same, and this modification in the Heroult process of steel refining is the chief difference of the Keller process. As the resistance is slightly increased by this method of heating, the power costs of the Keller process and furnace will be higher than in the case of the Heroult process, while on the other hand the consumption of electrode carbon will be lower. The Canadian Commissioners made a trial run with the Keller steel refining furnace at Livet, but owing to lack of time to complete the refining operation, no figures for the power consumption or quality of the finished steel were published in their report. A large steel refining furnace of this type has, however, been erected in the Steel works of Messrs. Holtzer & Co., at Unieux, France; the operation of this furnace will yield figures of exceptional interest and value to electro-metallurgists.

The furnace body weighs 50 tons, and rests on a steel cradle, which can be tipped for discharging purposes. A 1,500 h.p. steam-driven generating set will supply the electric current of 20,000 amperes at 60 volts. An output of between 24 and 32 tons of steel per day of 24 hours is expected from this plant. The iron used will be run into the refining furnace in the molten state from a Siemens-

Martin furnace, and in this way a considerable economy in electric energy will be attained.

The Kjellin Furnace and Process.—The Kjellin furnace differs from the three already described in the use of induced currents for heating the ore or metal contained in it. The furnace is in fact a transformer, with the ore or metal as secondary circuit. Only two other furnaces making use of this principle have attained practical trial—namely, the *Colby* furnace and the *Hiorth* furnace, a surprising fact in view of the undoubted advantages which result from this method of applying the electric current for metallurgical work.

Not only is the temperature kept within more reasonable limits than by arc-heating, but the heat is generated *within* the mass of the metal or ore, and not outside it, consequently there is less wear and tear upon the furnace linings and walls.

The greatest gain is, however, in the absence of electrodes, the cost of which form a considerable item in the running costs of the furnaces employing arc or resistance heating. Electrodes also lead to contamination of the products with carbon and silicon.

These advantages are of course balanced by one or two disadvantages, the most important of these being the non-adaptability of the furnace for refining impure raw materials. The temperature attained in the Kjellin furnace is not sufficiently high to eliminate the phosphorus and sulphur as completely as in the Heroult and Keller furnaces, consequently a certain selection of raw materials for refining by

MINING
U. OF C.
ASSOCIATI

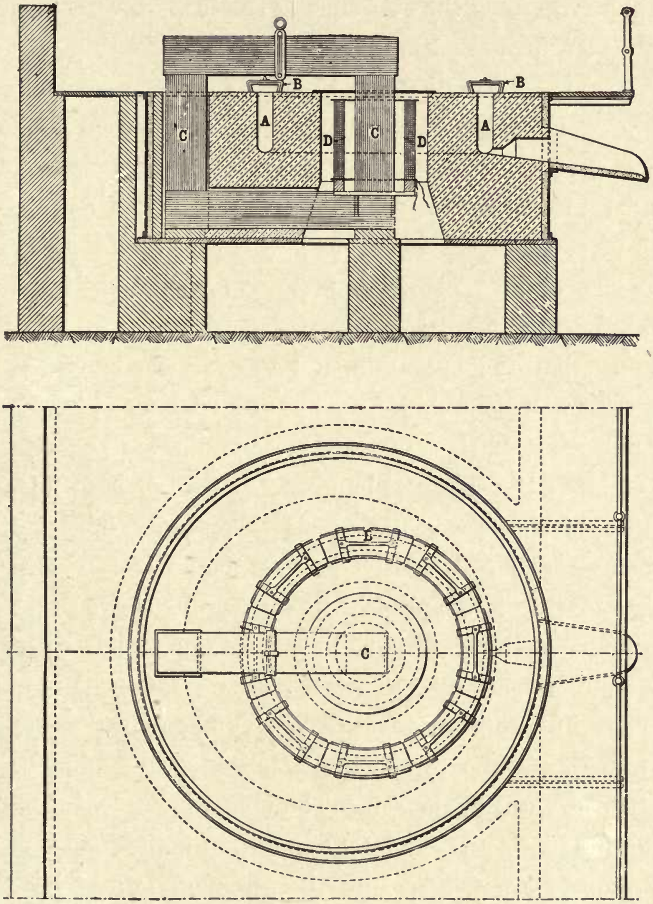


FIG. 45.—Kjellin furnace.

this type of furnace is necessary. A further disadvantage is caused by self-induction in the secondary circuit, since alternating currents are of course necessary. These electrical troubles diminish the efficiency of the furnace as compared with the arc and resistance furnaces of the direct type.

The earliest patent for this type of furnace was taken out by F. A. Kjellin, a Swedish engineer, in August, 1900, and since that year the furnace and process have been submitted to continuous trials at Gysinge, in Sweden. The first furnace erected was of 78 k.w. capacity, and produced 270 kgs. of steel per 24 hours. The size of the furnaces was gradually increased as experience was gained in their construction and operation, and the furnace employed by the Canadian Commissioners in their trial runs at Gysinge in February, 1904, was rated at 165 k.w., and produced 4,100 kgs. of steel per 24 hours.

Fig. 45 shows a sectional elevation and plan of this furnace. The core of the primary coil DD is shown at CC, and this is supplied with alternating current. The secondary circuit is formed by the ring of molten metal in the annular space AA.

The induced current in the secondary is roughly the number of amperes in the primary, multiplied by the number of turns contained in the coil. The furnace is operated in the following manner:—About 1,000 kgs. of molten pig-iron or scrap steel are poured into the annular ring AA of the furnace, and the current is switched on to the primary coil of the same. A current of 90 amperes at 3,000 volts is utilised; this is transformed in the secondary to a current of 30,000 amperes at 7 volts. 800 kgs. of cold pig-iron and scrap steel of the required composition, and in calculated proportions, are now added to the molten metal in AA, the covers BBB, which are made in short segments, being removed to allow of this addition taking place equally all round the ring. The covers are now replaced, and the heating is continued for 4 to 6 hours with different fluxes, until the metal has attained the degree of purity desired. The plug is then removed from the tapping hole, shown on the right of Fig. 45, and from 800 to 1,000 kgs. of the metal are run off and cast into ingots. The remaining 800 kgs. are left in the furnace to carry the current until fresh raw materials are added. It is found that there is always less carbon in the finished steel than is contained in the charged raw materials, while the silicon has increased. Two runs were made with this furnace in the presence of the Canadian Commissioners in February, 1904, and 1,985 kgs. of steel were obtained with a power expenditure of 1,851 k.w. in 12⅔ hours. The average power consumption was therefore 936 k.w. hours per ton of 2,204 lbs. The raw materials in each case were best Swedish pig-iron and Walloon bar iron, with scrap steel from previous charges. The steel produced contained only ·008 per cent. sulphur and ·010 per cent. phosphorus, with ·417 per cent. carbon in one case and 1·082 per cent. in the other.

The temperature of the steel when tapped is about 1,700° C. Using electric power at 40*s*. per e.h.p. year, the cost of producing steel by the Kjellin furnace and process is estimated at £7 per ton of 2,000 lbs., and the cost of erecting a 600 h.p. furnace is stated to be £830. The Swedish patents for the furnace are owned by the *Metallurgiske Patent Aktiebolag*, of Stockholm, and many licences to use the same have been granted. The German and Austrian patent rights have been sold to Messrs. Siemens and Halske, of Berlin. Krupp's have taken out a licence for the use of the furnace and process at their Essen works, and the Oberschlesische Eisen Industrie A.G. and the Poldihütte have also decided to adopt the process.

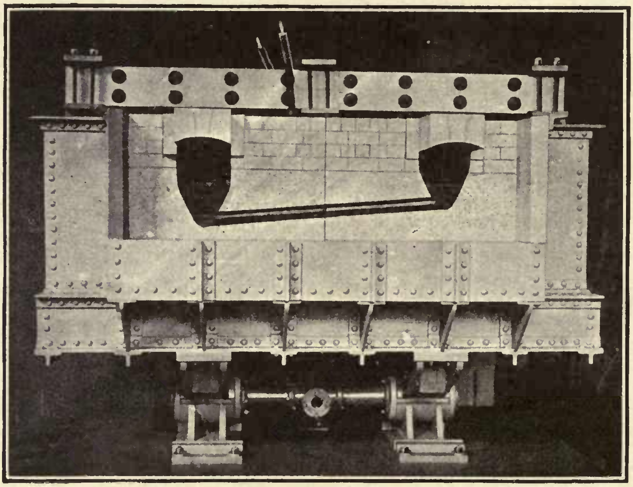


FIG. 46.—Gin furnace.

A project is under discussion in Stockholm for the development of the Trölhatta waterfall in the province of Norrland, where extensive deposits of iron ore are found, the proposal being to establish a large iron industry here with the aid of the Kjellin furnace and process. An experimental furnace of the Kjellin type has been installed in the metallurgical laboratory of the University at Sheffield, in order that Sheffield manufacturers may have an opportunity to become acquainted with the design and operation of the furnace. The steel-making firm of Vicars, at Sheffield, have also erected a Kjellin furnace in their works for experimental purposes.

The Gin Furnace and Process.—This furnace is the invention of M. Gin, a French electro-metallurgist. The patents for it are now owned by the *Société des Procedes Gin pour la Métallurgie Électrique*, a company floated in 1905 with a capital of 900,000 francs, to take over and exploit the Gin patents relating to all branches of electro-metallurgy. Gin has patented a large number of furnace designs, and only the latest of these can be described in this chapter. This is known as the combination tilting type of furnace. It is shown in elevation in Fig. 46. The furnace consists of a steel-framed brick chamber mounted on rocking bars, divided internally into three compartments: (1) a melting and oxidising chamber; (2) a de-oxidising and re-carburetting chamber; and (3) a final mixing chamber. These three chambers communicate internally by narrow connecting chambers. The electrodes enter the furnace by openings in the top, and are omitted from Fig. 46. The furnace is operated as follows: Molten iron is placed in chambers 2 and 3, and chamber 1 is charged with cold scrap and pig and the necessary fluxes. The electrodes of chamber 1 are connected to one of the current terminals, and the electrodes of chambers 2 and 3 are connected in parallel to the other terminal. The current passes from the electrodes to the metal through the slags in chambers 2 and 3, and then by means of the metal in the connecting channels back to the electrodes in chamber 1. The passage of the metal from one chamber to another is effected by tilting the furnace first on one side and then on the other. The channel between chambers 2 and 3 is so placed that it is uncovered when the furnace is tilted at the angle for tapping; in this way only chamber 3 is emptied. By tilting the furnace in the opposite direction the metal in chamber 2 is transferred to chamber 3, and the transfer of the metal from chamber 1 to chamber 2 is effected by a reverse movement. The slag is retained in chamber 1 in every position of the furnace. The electrodes dip into the slag but not into the metal in chamber 1, and the heating of the raw materials is therefore effected by the same means as in the Heroult steel refining furnace. This similarity is likely to raise important questions of patent validity at a later date. No figures have been published based on actual trials of this furnace, but the inventor has estimated that one ton of steel could be produced with a power consumption of between 640 and 720 k.w. hours. With iron ore at 15 francs per ton and electric power at 80 francs per k.w. year, the cost on this basis works out to between 82 and 83 francs per ton of finished steel. The Gin furnace and process are undergoing trial at Plettenburg in Westphalia.

The Girod Furnace and Process.—The Girod furnace for steel refining has been patented by M. Paul Girod, whose resistance type of furnace for the production of ferro-alloys has been described in Chapter VII. (p. 149). The steel refining furnace is shown in sectional elevation in Fig. 35 (p. 150). It is cylindrical in shape, is mounted on trunnions to facilitate tipping for discharge, and is lined with magnesia brick. The cover is distinct from the furnace, and is not shown in Fig. 35. It is made of silica brick; the upper carbon electrodes, four in number, pass through this cover into the furnace. The bottom of the furnace is formed of a number of water-cooled plates of cast-iron embedded in the brickwork. These plates connect with the materials in the furnace by means of narrow channels through the brickwork and function as the lower electrodes of the furnace. The channels are filled with molten iron before the furnace run is commenced, in order to provide electrical contact between the furnace contents and the lower electrodes. The furnace is operated as follows: The charge of cold scrap-steel and pig is placed in the furnace, with suitable fluxes, the cover with the upper electrodes in position is placed on the furnace, and the current is switched on. When the charge is melted, the upper electrodes are lowered until they dip beneath the slag, and the temperature is raised to the highest point possible, to eliminate the impurities of the raw iron. The steel is then carburised by adding a calculated weight of high carbon metal to the contents of the furnace. A 250 k.w. furnace of this type has been operated at Albertville, and has produced 1 ton of steel from 1½ ton charges in 4½ hours. The power consumption is stated to be 1,060 k.w. hours, and the cost of production, with electric power at ¼*d.* per k.w. hour, is estimated to be 29*s.* 7*d.* per ton of steel.

The Girod furnace has not been operated by any independent firm, so far as the writer is aware, and the manufacture of steel by it at the Albertville works is not carried on regularly.

III. Yields and Costs.

The figures used in the following pages for calculation of the electric power required to yield one ton of iron or steel by each of the various processes are drawn from the most reliable results or trials of which details have been published.[1] Where more than one set of figures is available, the mean or average of these is taken. The consumption of electric power is of course highest where the materials are charged cold into the furnace, and the aim is to produce iron or steel from suitable ores and slagging materials, while it is lowest where the electric current is used only for the final heating and purifying treatment of molten iron or steel from an open hearth, Wellmann or similar furnace.

The following are the figures, expressed as k.w. hours per ton of 2,000 lbs. :—

---

[1] Chiefly from the Report of the Canadian Expert Commission.

The Heroult Process.

1. Steel from molten open hearth steel	329
2. Steel from cold steel and scrap	864
3. Ferro-nickel pig from ore (cold)	2,342
4. Grey pig-iron from ore (cold)	2,693

The Keller Process.

1. Steel from molten pig-iron	631
2. Steel from pig charged cold	730
3. Grey pig from ore, etc., charged cold	2,292
4. Steel from ore, etc., charged cold	2,800

The Kjellin Process.

1. Steel from pig and scrap charged cold	866

The Stassano Process.

1. Steel from scrap charged cold	1,164
2. Steel from ore, etc., charged cold	2,804

The Girod Process.

1. Steel from scrap charged cold	1,136

The Gin Process.

1. Steel from molten pig-iron	618

Grouping the figures for each product and noting whether the materials are charged into the furnace cold or in the molten state, we have the following:—

Steel from scrap (*charged hot*).—Heroult, 329; Keller, 631; Gin, 618.

Steel from scrap and pig (*charged cold*).—Keller, 730; Heroult, 864; Kjellin, 866; Girod, 1,136; Stassano, 1,164.

Nickel pig from ores (*charged cold*).—Heroult, 2,342.

Pig from ore, etc. (*charged cold*).—Keller, 2,292; Heroult, 2,693.

Steel from ore, etc. (*charged cold*).—Keller, 2,800; Stassano, 2,804.

These figures show that as regards the production of *steel* from steel scrap and pig-iron charged into the furnace cold, the Heroult, Keller and Kjellin furnaces and processes all require about 800 k.w. hours per ton of finished steel, while the Girod and Stassano furnaces require about 1,150 k.w. Should the furnace be charged with molten iron and previously-heated and raw materials, a large saving in electric power is of course effected, and the ton of steel can then be purified and finished with an expenditure of 329 k.w. hours in the Heroult furnace, and of 620 k.w. hours in the Keller and Gin furnaces. It is probable that the figures for the two last-named furnaces can be much reduced, since the electric power required to heat 1 ton of steel up to 2,000° C. lies between 300 and 400 k.w. hours, and at least 350 k.w. ought therefore to be saved when starting the refining operation with molten iron or steel.

The figures for the production of ordinary pig or nickel pig from the ores charged cold show greater agreement, and 1 ton of pig can be obtained in either the Heroult or Keller furnace with an expenditure of 2,300 k.w. hours.

This amount is increased to 2,800 k.w. hours when steel is required as the final product of the treatment.

Basing our estimates on these figures, we have the following general averages:—

	K.w. hours.
2,000 lbs. steel from molten metal . .	400
2,000 lbs. steel from cold scrap and pig .	800
2,000 lbs. pig from ore, etc., charged cold .	2,500
2,000 lbs. pig from ore, etc., charged hot .	2,000

While the use of molten metal for charging the refining furnace is now carried out in the recently erected Heroult furnaces at Remscheid and at Syracuse, the attempts to subject the raw materials used for the ore reduction processes to a preliminary heating have not been successful. During the trials of the ore reduction furnace at Sault Sainte Marie in 1906, Heroult sought to utilise the waste heat of the exit gases from the furnace for this purpose. Stoppages occurred, however, in consequence of the softening of the charge in the down shaft of the furnace, and in the further trials the experiments in this direction were abandoned. The difficulty of designing a furnace which allows of the recovery of some portion of the heat of the exit gases should not, however, prove insurmountable, and no doubt at some later date either M. Heroult or some other electro-metallurgist will again attack the problem. The utilisation of all waste heat is a *sine quâ non* of a successful electric iron-smelting process.

IV. Comparative Costs.

Many figures for the cost of operating the electric furnace methods of iron and steel production, described in this chapter, have been published by the various inventors, and others interested in the new processes. These figures are, however, exceedingly divergent, and are of little value for determining how far these processes will be able to compete with the old established methods of iron and steel production. The cost of electric power, the cost of labour, the cost of iron ore and other raw materials, and the cost of freight on the finished products, all vary within wide limits in different localities, and yet have all to be considered in calculating the total cost of production. With so many variables entering into the calculations, the estimated total costs are bound to show wide divergence, ranging from 22*s.* up to £6 16*s.* per ton of 2,000 lbs.

It is therefore wiser, in attempting to form an opinion as to the comparative costs of operating the new processes of iron or steel manufacture, to take the one item of electric power only, and to calculate the cost of this for the different products, on the basis of $10·00 (£2) per e.h.p. year; that is, at one half the cost of electric power at Niagara Falls. We then have the following costs:—

	K.w. hours.	*s.*	*d.*
1. Steel from molten scrap . .	400	2	6
2. Steel from cold scrap and pig iron	800	5	0

	K.w. hours.	*s.*	*d.*
3. Pig-iron from ore, etc., charged cold	2,500	15	8
4. Pig-iron from ore, etc., charged hot	2,000	12	6

The practical iron and steel maker with these figures before him will be able to judge of the prospects for the electric furnace methods of production in his own district, bearing in mind that the low cost of electric power used as basis in the above calculations is often met with abroad, but is seldom or never approached in the United Kingdom.

For the general reader, unacquainted with the cost of iron and steel production, the following facts will provide material upon which to base a judgment as to the future of the electric furnace methods of iron and steel production in the United Kingdom and in other centres of this great industry. In the manufacture of the best crucible steel from 2½ to 3 tons of coke at 15*s.* to 17*s.* per ton are consumed per ton of steel produced, and practically the whole of this is used for heating purposes.

As regards the production of pig-iron in the modern blast furnace, only 16 cwts. of coke are now required per ton of pig, and of this total 6·5 cwts. are required to reduce the oxide of iron to the metallic state.

The saving in coke by the substitution of the electric furnace for the blast furnace can therefore only amount to

9·5 cwts. per ton of pig, equivalent to a reduction in the cost of 8*s*. The prospects for the electric refining furnaces and processes are therefore much brighter than those for the electric smelting processes, and it is exactly in the direction indicated by this judgment that the present developments are occurring.

CHAPTER XI.

LEAD.

I. INTRODUCTION.

THE attempts to reduce lead from its ores by electrolytic processes have not so far been successful, and although one process of this kind was operated for some time upon an industrial scale at Niagara Falls, the works was finally closed down. At the present time the application of electric methods in the lead industry is confined to the refining operation. The Betts electrolytic refining process is in use at Trail in British Columbia and at Newcastle in England.

Betts recently has been experimenting upon the use of electricity as agent for the reduction of lead ores, but these trials do not appear to have proceeded beyond the laboratory stage; and it is yet too early to say whether a practical and economical process will result.

Details of the unsuccessful Salom ore reduction process, and of the Betts electrolytic refining process are given below, together with a *resumé* of Betts' later experiments. A few notes upon the electrolytic processes for manufacture of *white lead* are added in section IV.

II. THE SALOM LEAD REDUCTION PROCESS.

This process was patented by *P. G. Salom*, in America, and was worked from 1901 to 1904 by the Electrical Lead Reduction Company at Niagara Falls, with a plant utilising 500 h.p. The process was electrolytic in character, since the electric current was employed to generate hydrogen from an acid solution, and it was the action of this "nascent" hydrogen when liberated at the cathode, that reduced the lead sulphide ("galena") to the state of spongy lead. The galena was carefully washed to remove gangue and other impurities, and was then ground to a fine powder. This powdered ore was then fed into a large circular cell having a rotating bottom, and containing a dilute solution of sulphuric acid. As the rotating cell bottom carried the ore slowly away from the charging point and between the electrodes, the galena was subjected to the action of the nascent hydrogen liberated at the cathode plates of the cell. When one revolution of the bottom had been completed in 1½ hours, the sulphide was entirely reduced to spongy lead, and was removed from the cell by a fixed scraper placed at this point. The operation of the cell was entirely automatic, the rate of feed being continuous, and the spongy lead being discharged from the top of the cell by means of the fixed scraper. Each cell had a capacity of 200 lbs. of lead sponge per day, and the power consumption was stated to be ·5 e.h.p. hour per lb. of lead sponge. Difficulties were met with early in the development of this process, due to the escape of the highly poisonous and dangerous gas—sulphuretted hydrogen—from the cells, and modifications in the design of the cell were introduced at Niagara Falls. Seventeen of the new form of cell were erected at the works in 1904, and the trouble caused by escapes of SH_2 was minimised but not abated. Experiments directed towards the use of the liberated SH_2 for manufacture of sulphuric acid were proceeding when the Company went into liquidation, and the works was shut down.

The lead sponge produced at Niagara Falls by this process was converted into litharge by roasting, but it was hoped that it would prove valuable as raw material for the manufacture of white lead, and for storage batteries.

The electrical plant at the works consisted of 2 × 300 h.p. induction motors, direct coupled to 2 × 250 k.w. generators, producing direct current at 110 volts. The writer is not aware whether this works has been dismantled, or whether the inventor has hopes of restarting the plant under more favourable financial conditions.

III. The Betts Electrolytic Refining Process.

This process is the invention of Anson G. Betts, an American engineer, and it was first operated at Trail, British Columbia, about the year 1902. The process depends upon the use of raw lead or of lead bullion as anode material, in an electrolyte of lead fluo-silicate. This electrolyte is made by diluting commercial hydro-fluoric acid with water, and then dissolving powdered quartz in the mixture. Sheets of electro-deposited lead are used as cathodes. An e.m.f. of only ·2 volt suffices to transfer the lead from anode to cathode in this electrolyte, while the silver, antimony, bismuth and other impurities of the raw lead remain insoluble, and collect upon the floor of the vat as slimes. The original plant erected at Trail comprised 20 depositing vats, and the capacity was stated to be 10 tons of lead bullion per day. Some difficulty in operating the plant successfully was caused at first by "treeing" of the deposit, and by short circuiting of the anodes and cathodes. This has been overcome by adding a small amount of glue to the electrolyte.

The plant at Trail was altered and enlarged in 1903 and again in 1905. The tanks in the enlarged works are 7 feet 2 inches long by 2 feet 6 inches wide by 3 feet 6 inches deep, and are made of 2 inch planks, coated with an acid-proof paint. The electrolyte contains 10 per cent. of hydro-fluo-silicic acid with 5 per cent. of lead in solution; 2 lbs. of glue are added every other day to each 2,000 cubic feet of electrolyte. The anodes are cast 1 inch thick and are placed 4⅛ inches from centre to centre. Each tank when fully charged contains 20 anodes and 21 cathodes. The current density used is 15 amperes per square foot. The vats are arranged in cascade fashion, and the circulation of the electrolyte is obtained by means of 1½ inch hard rubber pipes. A rapid circulation is found to conduce to good deposits, and to absence of treeing.

The anodes weigh each 300 lbs., and require 6 days for their disintegration, 10 to 20 per cent. of scrap being obtained. The cathodes weigh from 150 to 200 lbs. when removed, and are washed and scrubbed before melting down, the antimony and tin that have been deposited with the lead being removed as *oxides* in the dross formed during this operation.

The slimes which collect on the floor of the vat and with the scrap are well washed, and are then treated for recovery of the silver, antimony and other metals. According to Haber, who visited the States in 1903, the best method of working up the slimes has not yet been settled upon. Lead refined by the Betts process is reported to have a purity of 99·99 per cent. and to contain only ·0005 per cent. copper, ·001 per cent. iron, and ·0006 per cent. antimony. The present output of the refining plant at Trail is 20 tons refined lead per day. The chief disadvantages of the Betts process are the large amount of lead-fluo-silicate absorbed by the slimes, and the large amount of antimony present in this residue, portion of this antimony being combined with the lead. However these disadvantages do not outweigh the convenience and economy of the method for refining lead bullion, and it is satisfactory to note that an English firm of lead-smelters, Messrs. Locke, Blackett & Co., have decided to erect an electrolytic refining plant, on the Betts system, at their works at Newcastle-upon-Tyne. No details of this plant, which is now at work, are available for publication.

IV. THE BETTS ELECTROLYTIC REDUCTION PROCESS.

The first process which was protected by U.S.A. Patent No. 816,764, was worked out in conjunction with W. Valentine, and depended upon the electrolysis of lead sulphide in a bath of fused lead chloride. This method failed, owing to the accumulation of impurities in the electrolytic bath.

The following process was then devised:—first stage, fusion of the ore and formation of a matte containing chiefly lead sulphide; second stage, electrolytic decomposition of the fused matte under such conditions that the iron remains fused as sulphide and does not contaminate the reduced lead or clog the baths. It was found that the use of molten sodium chloride with the lead sulphide, as electrolyte, gave the conditions required to obtain these results.

This process had not been tried experimentally in May, 1906, but the probable costs of operation had been calculated as follows, from calorific data. The first portion of process would require 639,000 calories of heat per ton of ore; and the cost would vary between $2·44 and $3·48 per ton—according as the cost of electric power was $20·00 or $40·00 per e.h.p. year.

The electrolytic portion of the process would add $2·68 or $3·83 to the cost, making the total costs either $5·12 or $7·31 according to the cost of power. Deducting the value of the recovered sulphur, the net cost of the electrolytic reduction process is $2·97 or $5·16, and in the opinion of Betts, the process is one which can compete with existing smelting methods, especially as the losses by volatilisation will be lower.

V. ELECTROLYTIC PROCESSES FOR WHITE LEAD PRODUCTION.

White lead is a mixture of lead carbonate and lead hydrate, and as the ordinary acetic acid method of manufacture by the Dutch system is dirty, lengthy and troublesome, numerous attempts have been made to utilise electrolytic methods in this industry. Up to the present date no entirely satisfactory method has been evolved, but it is quite possible that ultimately the difficulties attending the solution of the problem will be overcome. *Richards and Roepper*, in U.S.A. Patent No. 644,791, suggest the electrolysis of a mixed solution of sodium sulphite and sodium acetate, with lead electrodes, and a current density of between 100 and 200 amperes per square metre. The CO_2 produced under these conditions at the anode unites with the lead hydroxide and forms white lead. The *Luckow* process is a German one, and is based upon the use of a 1½ per cent. solution of sodium chlorate and sodium carbonate as electrolyte, while that of *Oettli* is based upon the use of sodium chlorate solution as electrolyte, with reversal of the current direction every five minutes, and introduction of CO_2 into the electrolyte from some external source. The two last named processes have been exploited in Germany upon an industrial scale, but no trustworthy information can be obtained as to their present position or success.

CHAPTER XII.

MISCELLANEOUS PRODUCTS.

I. INTRODUCTION.

IN this chapter a group of minor electro-metallurgical products, which are now being manufactured upon a small industrial scale at various works in Europe and America, will be dealt with. In some cases the manufacture of these products is carried on in conjunction with the manufacture of other electro-metallurgical or electro-chemical products, as for instance the manufacture of *Calcium* by the Bitterfeld Elektro-chemishe Werke in Germany, which produces mainly alkalies and chlorine in addition to chromium, sodium, and potassium. In other cases the manufacture is carried on alone, as that of *carbon bisulphide* at Penn Yam in America.

The following manufactures will be described:—Calcium, carbon disulphide, carbon tetra-chloride, diamantine, magnesium, phosphorus, silicon and its compounds. The two carbon compounds are not strictly electro-metallurgical products, but since the furnaces used in the manufacture of these compounds are electro-thermal in principle, it has been thought well to include a description of them in this volume.

II. Calcium.

The metal *calcium* is one of the latest additions to the ranks of the commercial metals, for until three years ago it had not been produced, except in very small quantities for laboratory investigation. The early attempts to produce the metal in larger quantities by electrolysis of the fused chloride failed. Not until Dr. Rathenau, of the Elektro-chemische Werke, Bitterfeld, exhibited large sticks of metallic calcium at the Bonn meeting of the German Society of Electrochemists in 1904 was it known that the special difficulties met with in the production of this metal, by electrolysis of its fused chloride, had been overcome. The difficulties are due to the fact that the calcium when melted, is soluble in the molten electrolyte, while if it is attempted to carry out the separation of the metal at a temperature below its melting point, it is deposited at the cathode in a spongy form. Messrs. Suter and Redlich, of the Bitterfeld Works, have overcome this difficulty by using for the decomposition of the bath an electrode which is placed vertically with its lower end just touching the surface of the electrolyte. When a current is passed through the electrolyte using this vertical electrode as cathode, a button of metallic calcium is formed upon the end of it. The vertical cathode is then subjected to an upward movement of $\frac{1}{8}$ or $\frac{1}{4}$ inch, which leaves only the under surface of this button in contact with the bath. A new button of metallic calcium is formed, and the upward movement or jerk is again repeated. In this way the metal is removed as formed, from contact with the molten electrolyte, and an irregular stick of metallic calcium 12 or more inches in length by ½ inch in diameter, weighing 1½ to 2 lbs., is produced. The metal as seen in these sticks is dark grey in colour, and has a greasy surface. If brought into contact with water, it forms the hydrate and liberates hydrogen. Its sp. gr. lies between 1·51 and 1·54, and its melting point is about 800° C. The metal can be obtained in this way without difficulty from a bath of the molten chloride, and it is not necessary to mix calcium fluoride (fluor spar) with the electrolyte in order to obtain good results, although in some cases the addition of fluor spar has been found advantageous. The bath should be charged with fresh neutral calcium chloride, and maintained at a temperature of 675° C. This method is protected by German Patent No. 155,443 of 1903, and it has been in use at Bitterfeld since that year. The output of the metal is small, since no wide field of use has yet been found for it, although it can be now placed on the market at the low price of 6*s*. per cwt.

Calcium in some of its chemical properties resembles aluminium, for it has a great affinity for oxygen. The attempts to use it, however, in place of this metal for clearing iron and steel of their gaseous impurities, before casting, have not so far yielded satisfactory results. As a substitute for aluminium in the Goldschmidt *Thermit* processes, calcium may also be used, but here again there are difficulties in its application which have not yet been surmounted. None of the alloys of calcium with other metals are of any practical value, brittleness always being produced by the addition of the lighter metal, and it is partly due to this fact that the industrial possibilities of the new metal seem to be greatly restricted.

III. Carbon Bisulphide.

This chemical has been produced since 1902 by an electrothermal process at Penn Yam, New York. Electricity is here applied in place of solid or gaseous fuel, to furnish the heat required to bring about the combination of the carbon and sulphur used as raw materials of the manufacture. The furnace and process are the invention of Mr. E. R. Taylor, of Penn Yam. The furnace is shown in sectional elevation in Fig. 47. It is constructed with an iron shell, and a chambered lining of firebrick extending up to the top. The plates through which the electrodes (C D) pass are insulated with mica and asbestos. The carbon is placed in the bottom of the working chamber, fragments of broken coke are placed around

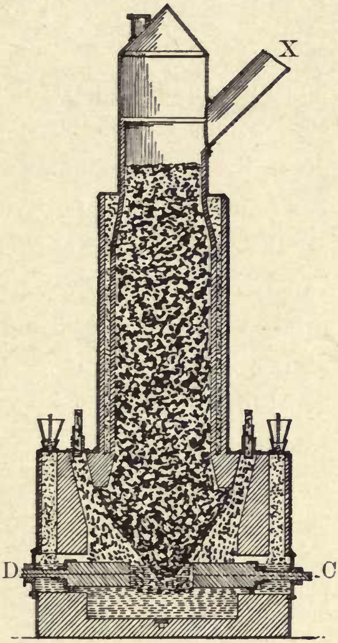


Fig. 47.—Taylor's carbon-disulphide furnace.

and between the two electrodes, and the body of the furnace is filled with charcoal, as shown in the figure. The annular spaces left in the lower part of the furnace are now filled with sulphur, and the current is switched on.

As the heat generated by the carbon resistance between the two electrodes spreads, the sulphur is melted and runs down to the lower part of the working chamber. Should the current be too large and the heat generated be too great, this chamber fills with melted sulphur, and as this is a bad conductor of electricity, it acts as an automatic regulator for the current passing between C and D. The sulphur is vaporised by the heat generated in this portion of the furnace, rises, and passing up the shaft into the hot charcoal, unites with this to form carbon bisulphide vapour, which finds an outlet by the exit pipe X. The gas passes from X into the cooling and condensing plant, where it is condensed to a liquid form. The loss of heat by radiation from the lower part of the furnace is minimised by constructing it with several annular rings, all of which are filled with powdered sulphur.

The original furnace erected at Penn Yam was 41 feet in height and 16 feet in diameter, and utilised a current of 4,000 amperes at 50 volts. It produced 8,000 lbs. of carbon bisulphide per 24 hours. The author is unable to state whether the extension of the plant contemplated in 1903 has yet been carried out.

Carbon bisulphide is a chemical used as a solvent for gums, caoutchouc, phosphorus and sulphur, and the demand for it is therefore somewhat limited. It is also used in America as a germicide and for killing vermin, and in France for the manufacture of artificial silk from cellulose. The total consumption is, however, small, and it is obvious, therefore, that a process which permits its continuous production at the rate of 8,000 lbs. per 24 hours will meet with some difficulties due to over-production. If more than one works in this country takes up this manufacture, the price of the product may be forced down to a point which ceases to be remunerative; and so far as the author is aware, the factory at Penn Yam is the only one manufacturing the product on a large scale by aid of electric heat.

IV. Carbon Tetra-Chloride.

This compound has been produced upon an industrial scale by F. G. Machalske in an electric furnace at a small works at Brooklyn, New York.

The method depends upon the use of a compound of chlorine, of a reagent which will react with the base of this compound when the chlorine is freed by heat, and of some form of carbon. Coke, quartz sand and sodium chloride are the most suitable raw materials. When these three materials are heated together to a temperature of nearly 2,000° C. the chloride is freed from its previous combination and unites with the glowing carbon to form carbon tetra-chloride, which passes off as a gas to the condensing apparatus, while sodium silicate or soluble glass remains behind in the furnace. The furnace in which this reaction is carried out is shown in section in Fig. 48, carbon electrodes being employed to lead the current into and away from the interior of the furnace. An electric current of 500 amperes at 60 volts suffices to produce carbon tetra-chloride by this furnace and method, the chief conditions for success being that no oxygen or air should be present in the furnace after the materials have attained a temperature of 1,000° C., and that a very efficient condensing apparatus be attached to the furnace plant. Furnaces of 150 k.w. and 300 k.w. capacity have been employed by Machalske for this manufacture at Brooklyn. The author is not aware whether the production of carbon tetra-chloride is being continued at this works by the electro-thermal process. The compound is one for which there is only a limited use, and the proposal to employ it instead of petroleum spirit or benzol in "dry-cleaning" works has not met with great favour, on account of the possible effects of its vapour upon the health of the

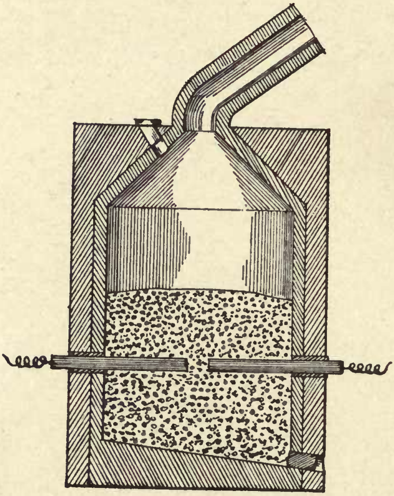


FIG. 48.—Machalske carbon tetrachloride furnace.

employés. Carbon tetra-chloride is also at present being manufactured as a by-product in many works by various processes, notably in several of the electrolytic alkali works, by simply passing the chlorine from the cells over glowing coke. The price of the compound is therefore likely to remain low until a very large demand for it arises. Since it is an excellent solvent for oils, fats and grease, it is possible that this large demand may be created, and that the Machalske electric furnace method of production may find a wide field of usefulness.

V. Diamantine.

Diamantine is the trade-name given to a new product, obtained by heating alumina, with small quantities of silica, to a high temperature in an electric furnace. The compound is therefore a silicide of aluminium. When finely ground and mixed with clay and water, this new material is said to form a useful wash for the inside linings or walls of furnaces exposed to high temperatures. This new refractory material is now being manufactured upon an extensive scale by the *Diamantine Werke* at Rheinfelden, in Germany.

VI. Magnesium.

This metal is being produced in Germany by electrolysis of *Carnallite* (a naturally occurring salt containing potassium and magnesium chlorides), with separation and recovery of the chlorine liberated at the anode. The cell designed for this operation by Haag is divided into three sections. The upper portion receives the raw carnallite, and in this it is melted by resistance heating. The central portion of the cell is the electrolysing chamber; in this the molten carnallite is decomposed by the electric current using perforated carbon electrodes. The third and lowest portion of the cell receives the molten magnesium and undecomposed portions of the carnallite, and by an ingenious device these are here separated and discharged by different outlets. The chlorine gas rises through the hollow and perforated carbon anode, and passes away by the funnel-shaped hood which covers the upper portion of the cell. Both the raw materials and the final product are kept in the molten state within the cell by resistance heating, and the various portions of the cell are kept gas tight by liquid seals, formed of these molten materials. The whole cell is enclosed in an air chamber to minimise the heat losses due to radiation. The process is continuous, and heat may be applied externally if desired, to assist in maintaining the desired temperature.

VII. Phosphorus.

A large proportion of the phosphorus production of the world is now manufactured by electro-thermal processes, but details are difficult to obtain, since the phosphorus industry is almost a monopoly in each country in which it is carried on. In the United Kingdom Messrs. Albright and Wilson, of Oldbury, control the manufacture.

Bone ash is the raw material of nearly all processes for the production of phosphorus. In the electro-thermic process this is mixed with silicic acid and carbon, and the mixture is then heated to a temperature of between 1,300° C. and 1,500° C., between which limits the following reaction occurs:—

$$Ca_3\,P_2\,O_8 + 3\;SiO_2 + 5\;C = 3\;Ca\;SiO_3 + 5\;CO + 2\;P.$$

Bone ash with sand and carbon yield calcium silicate, carbon monoxide gas and phosphorus. The phosphorus commences to distil over at 1,150° C., and is all expelled from the mixture before a temperature of 1,450° C. is passed.

The plant used in Germany for carrying out this method of manufacture, according to Hempel, consists of a gas-tight iron cylinder, lined with fire-clay. In the lower portion of this cylinder, thick carbon electrodes are introduced. The process is a continuous one, the molten calcium silicate being run off without intermission, and the temperature of the whole furnace being maintained at a point above that at which phosphorus condenses or solidifies. The phosphorus obtained by this method is much cleaner and purer than that obtained by the older sulphuric acid method of manufacture, and a yield of 92 per cent. has been obtained. The use of silicic acid in phosphorus production is stated by Hempel to be only possible with electric furnace methods of manufacture, since the high temperature required inside the retort is not attainable by other methods of heating.

F. J. Machalske, of Brooklyn, has produced phosphorus by a method similar to that described above, the raw material in this case being the ordinary phosphate rock used in America as a fertiliser.

The furnaces used by Machalske have an internal chamber 12 inches by 18 inches, with a carbon bottom, and sides of calcined magnesia, the cover being made of fire-clay and red brick. Two electrodes, each 8 feet in length by 4 inches in diameter, are passed through holes in the cover. The electric current passes between these electrodes and the base of the furnace, melts the charge, and causes the phosphorus to distil over. The phosphorus vapour is condensed under water, and the slag runs off from side vents, the process being a continuous one. With electric power at 3 cents per e.h.p. hour, Machalske claims that yellow phosphorus can be produced at a total cost of 7 cents per lb. by this method. A company called the *Anglo-American Chemical Company* was formed in 1901 to purchase the patents and work the Machalske process and furnace upon a commercial scale, but the author is unable to say whether this company is still in existence, or is now producing phosphorus.

VIII. Silicon and its Compounds.

Silica and alumina are the two most commonly occurring oxides in the earth's crust, the first named representing over 55 per cent. of the earth's total weight. From this fact one might draw the conclusion that these two elements have the greatest affinity for oxygen of all the metals and elements found near the earth's surface. This deduction is correct, for it is supported by the results of chemical investigation. According to Ostwald, the heat of formation of all the silicon compounds is still very uncertain, but since this element can rob both iron and copper of their combined oxygen when introduced into a bath of molten metal containing the oxides of these two metals, the heat of formation of silicon oxide or silicic acid must be exceptionally large.

Silicon has in fact a very high affinity for oxygen, and in the form of silicon-copper and ferro-silicon it has already attained a large sale for metallurgical purposes. The manufacture of carborundum and of ferro-silicon has already been described in Chapters V. and VII. In this section the production of *metallic silicon*, of *silicon-copper*, and of *siloxicon* will be dealt with.

Metallic Silicon.—F. J. Tone has prepared metallic silicon in large quantities at Niagara Falls, by heating a mixture of sand and pulverised coke in an electric furnace to a high temperature. The mixture must be in the molecular proportions of 1 of silica to 2 of carbon. The temperature of the furnace requires careful regulation in order to avoid the formation of siloxicon, carborundum, or graphite. The furnace used by Tone consisted of a vertical and cylindrical chamber, built of firebricks and open at the top. The reaction proceeded according to the equation:

$$SiO_2 + 2\,C = Si + 2\,CO.$$

(Sand and carbon yield silicon and carbon monoxide.)

The heat is produced by the resistance of the charge to the passage of the electric current. As the reduction proceeds the silicon falls to the bottom of the furnace, where it is tapped and cast into pigs. The metal so obtained is dark in colour, and crystalline in structure.

Its density is 2·34, and its melting point 1,430° C. The commercial product contains over 95 per cent. silicon, and the impurities can be easily reduced in amount by using purer raw materials in the manufacture. As in the case of *calcium*, the manufacture of *silicon* in large quantities waits upon the growth of the demand for the metal, and up to the present time no large field of usefulness has been opened up for this latest addition to the commercial metals.

Siloxicon.—Siloxicon is a very refractory compound containing silicon, oxygen and carbon, in the proportions represented by the chemical formula, $Si_2\,C_2\,O$. It was discovered and patented by Acheson, and is made by heating together finely powdered coke, sand and sawdust to a temperature of 2,600° C. in an electric furnace. The furnaces used in this manufacture are similar to those used for the manufacture of *Carborundum* (see Chapter V.); but the temperature is kept lower than for the latter compound.

As an example of the mixture and current used, $33\frac{1}{3}$ parts powdered coke were heated with $66\frac{2}{3}$ parts sand and a small amount of sawdust, using a current of 500 to 600 amperes at 85 volts for 9 hours. The product of the reaction has a sp. gr. of 2·73, and is greyish-green in colour. It is reduced to powdered form before placing on the market.

Siloxicon is a self-binding material, and when moistened with water it can be moulded to any desired form and then fired. It has been used for making bricks, crucibles, retorts and blocks for furnace linings. In some instances tar, asphaltum, water, glass and pitch have been used as binding agents, in the manufacture of blocks or vessels from siloxicon. A semi-plastic mixture of this kind may be employed for tamping the interior of furnaces designed for high temperature reactions. In neutral or reducing gases, siloxicon is unaffected until its temperature of decomposition is reached (3,000° C.), but in the presence of oxygen or of alkalies it decomposes at 1,500° C. In each case oxygen is liberated and carborundum is produced. This fact somewhat restricts its field of usefulness as a refractory material for electric and other furnaces. A company was floated in America in 1904, called the *Acheson Siloxicon Article Company*, to take up and develop the manufacture of bricks, crucibles, retorts, etc., from this new refractory material, and the erection of a works at Niagara Falls was contemplated. The author is unable to state how far this scheme has been realised.

Silicon-copper.—Silicon-copper is an alloy of silicon and copper which is used for clearing molten copper from its free and combined oxygen. Owing to its high affinity for the latter element, the silicon acts as a purifying agent and thus produces sounder castings. The applications of silicon-copper in the copper foundry are similar to those of ferro-silicon in the iron and steel foundry, and the manufacture of silicon-copper is therefore one of the minor electric furnace industries.

The chief producers of silicon-copper are the firm of Cowles Brothers at Lockport, N.Y.; and the alloy is also reported to be manufactured by the Carbide Works at Landeck and Mattrei in Europe. No details of the method of manufacture or of the furnace used are available for publication.

CHAPTER XIII.

NICKEL.

I. INTRODUCTION.

THE remarkable effect produced by nickel upon the physical properties of iron and steel, when alloyed with these in proportions varying from 2 to 14 per cent., has raised the first named metal to a position of great importance in the metallurgical industries.

The ternary and quaternary steels, containing nickel as their third or fourth component, include some of the strongest alloys yet made ; details relating to the quaternary steels have been already given in Chapter VII.

As regards ternary steels, the increased tenacity conferred upon steel by the addition of nickel is shown by the following figures for the tests of two bars each 1 inch in sectional area, the one made from the unalloyed steel, and the other made from steel containing 11½ per cent. of nickel. The latter possessed a tensile strength of 89 tons, as compared with 32 tons for the bar of unalloyed steel. The resisting power of nickel steel to compression and shock is also much greater than that of the unalloyed steel, and nickel steel is now very generally employed for the guns of battleships. German gun-makers are reported to use steel containing

5 per cent. of nickel; English makers use steel containing up to 4 per cent. of nickel.

The metallurgy of nickel is therefore of considerable importance for all steel manufacturers. In this chapter details will be given of the various processes which have been tried upon an industrial scale for the extraction or refining of the metal by electrolytic or electro-thermal methods. Although the trials have extended over twenty years, and are still being continued, the application of electrical methods in the nickel industry is confined to a few works in Europe and America, and the greater proportion of the output of the metal is produced by the older metallurgical processes. This is due, not to the inefficiency of the electrolytic and electro-thermal processes, but to the facts that a metal testing 99·2 per cent. purity can be easily produced by the older methods, and that there is practically little demand for a purer metal, the chief demand being by the manufacturers of German silver. The nickel used in the iron and steel industries is not pure, but is supplied in the form of an alloy—*Ferro-nickel.* The electric furnace is no doubt most suited for producing this alloy, although up to the present the demand for ferro-nickel appears to have been met by the ordinary metallurgical processes.

At the present time nickel is produced by three companies by electrolytic or wet methods. At Sault Sainte Marie, in Canada, a plant for the production of ferro-nickel by the Heroult process and furnace is in course of erection, experimental trials made in 1906 having shown that the ores of this district can be successfully smelted by electro-thermal methods. Details of these four processes, and of others which have failed, will now be given.

II. Wet Processes.

As already stated in the introduction to this chapter, the wet processes for refining nickel are electrolytic in character, and depend upon the application of the electric current for separating the metal from solutions of its salts. The electro-deposition of nickel is comparatively simple when pure solutions of the salts are used, and when only a thin coating of the metal upon the cathode is desired. "Nickel-plating" under these conditions is now an important branch of the electro-plating industry, and is very extensively practised in the two head-quarters of the cycle manufacture:—Coventry and Birmingham. The simplicity, however, disappears when impure solutions containing copper, iron and other metals have to be dealt with; and this impurity of the solutions obtained from the nickel matte has been one of the chief difficulties met with in attempting to apply electrolytic methods for refining the metal.

The theoretical amount of energy required to deposit 1 kg. of nickel from solutions of its salts depends upon the state of oxidation in which the metal is present, and upon the nature of the anode and cathode.

Using insoluble electrodes, and the salts of the higher oxide of nickel, 2·53 k.w. hours are required to deposit 1 kg. of the metal from the chloride solution, and 2·87 k.w. hours from the sulphate solution. As a rule soluble anodes are used, however, and these figures are reduced owing to the e.m.f. created by the solution of nickel and copper at the anode. Thus Ronco states that at Cleveland 1·71 k.w. hours suffice to deposit 1 kg. of nickel. Taking the cost of the k.w. hour at ½*d*., the cost of current is seen to be less than 7 per cent. of the value of the nickel obtained. The cost of the electrical energy required is therefore a comparatively unimportant item in the total cost of nickel production, and as the wet methods of treatment allow the platinum and palladium to be recovered from the anode sludge, it would seem certain that in time all ores containing these precious metals will be subjected to the combined "dry" and "wet" methods of treatment. As in copper refining, the value of the "impurities" of the nickel ore may exercise a very important influence upon the development of the electrolytic processes.

The difficulties attending the electro-deposition of nickel are, however, greater than in the case of copper. The anode material always contains a larger percentage of impurities, and, after the deposit of nickel at the cathode has reached a certain thickness, it shows a disposition to peel away and detach itself in flakes.

Foerster has investigated the latter phenomenon, and has proved that it can be overcome in the case of sulphate solutions by working them at a temperature of 60° C.[1] The deposition of thick plates upon a practical working scale of operations is therefore feasible, and the only special difficulty is that due to the losses arising from anode scraps and slime, which, according to Ulke, in some cases rise as high as 40 per cent. of the alloy used. Of course this is re-worked, but the tendency of the anodes to crumble away adds greatly to the expense of the process.

---

[1] "Zeits f. Elektrochemie," Vol. IV., p. 160.

The Thum Process at the Balbach Company's Works, Newark, U.S.A.—This company was, the writer believes, the first to obtain plates of nickel of marketable thickness by electro-deposition, the process used having been in operation at the Newark Works since 1894. The raw material used was formerly obtained from the Orford Copper Company, and contained 95 to 96 per cent. nickel, with the impurities already enumerated, and ·50 oz. platinum per ton.

The crude nickel was cast into anode plates measuring about 20 inches by 30 inches, by the Balbach Company, and was suspended in a solution of nickel sulphate, kept either neutral or slightly alkaline with ammonia. Plates of rolled nickel were used as cathodes, and, under suitable conditions as regards temperature and current density, plates of nickel 20 inches by 30 inches by $\frac{3}{8}$ inch thick were obtained. The deposited metal tested as follows:—99 to 99·6 per cent. nickel; ·12 to ·14 per cent. cobalt; ·10 to ·20 per cent. copper; ·10 to ·40 per cent. iron; ·03 per cent. arsenic, with traces of manganese, sulphur, silicon and platinum.[1] Using this process, the Balbach

---

[1] Ulke, *Engineering Magazine*, U.S.A., Dec., 1898.

Company have produced in the past, at Newark, three tons of nickel per day, and the metal obtained is stated to have been satisfactory as regards toughness and tensile strength.

The same process was stated to have been in use at the Nickel Extraction Works of Gustav Menne & Co., at Siegen, in the year 1896.

The Orford Copper Company's Process, Bayonne Works, New Jersey, U.S.A.—This company has only recently taken up the production of pure nickel by electrolysis. It is reported to be using a process in which a solution of nickel chloride is employed as electrolyte. The cathodes are thin sheets of pure nickel measuring 36 inches by 48 inches, and are coated with graphite. The anodes are made from the nickel matte, obtained by the old "tops and bottoms" process of separating copper and nickel sulphides. In this process the roasted ore is subjected to repeated smeltings with sodium sulphate and coke in a cupola furnace; the nickel sulphide, being heavier than the iron and copper sulphides, accumulates in the lower portion of the cooled mass.

The use of this impure nickel sulphide as anode material, in a vat containing nickel chloride as electrolyte, leads to solution of the nickel and liberation of the sulphur in the free state at the surface of the anode. No information is available showing how this free sulphur is dealt with in the Orford Company's works. The electro-deposited plates obtained by this process are so tough that they require cutting by means of power-shears. The surface is " wart-like."

The " Browne " Process at the Cleveland (Ohio) Works of the Canadian Copper Company.—The Browne process of nickel refining was in use by the Canadian Copper Company until the Cleveland works were taken over by the *Nickel Trust Company*, and it is stated that the process is to be used again when the installation of electric power plant of the Canadian Copper Company at Copper Cliff is completed. The Browne process is cyclic in character, and depends upon the use of a mixed copper and nickel chloride solution as electrolyte. Anodes of a copper-nickel alloy and cathodes of pure copper are employed in the vats, and the copper is first deposited. The remaining copper and any iron present in the solution are finally removed by chemical means, and the neutral pure solution of nickel chloride which remains after this treatment is then electrolysed with carbon anodes and cathodes of pure sheet nickel. The electrolyte is kept hot during the deposition of the nickel. The solution of copper and nickel chlorides for the vats is maintained at the right concentration by passing portion of the electrolyte from the anode compartment of the copper depositing vats through a tower in which it comes into contact with the Bessemerised matte, and also with the chlorine gas from the closed anode compartment of the nickel depositing vats. Special attention must be given to this portion of the process, and to the freedom of the electrolyte from other impurities.

According to Ulke, the Cleveland works in 1902—3 were producing from 8 to 15 tons of nickel per month by the Browne process; and it was only discontinued because the operation of the Balbach process was found to be the more economical at that date.

The Hybinette Process at the Sault Sainte Marie Works of the Canadian Nickel Company.—This process is one of the most recent for the wet treatment of nickel ores, and it is about to receive trial at Sault Sainte Marie, at the new works of the Canadian Nickel Company. The process, like the Browne process, is cyclic in character, but a sulphate solution is employed in place of the chloride, and the cell used for the electro-deposition of the nickel is somewhat complicated in design and structure (Fig. 49). The electrolyte employed is a dilute solution of nickel sulphate, to which a small quantity of boric or phosphoric acid has been added. The anodes are cast from a ferro-nickel-copper alloy, which is obtained by direct smelting of the ferro-nickel ores mined in the district around Sault Sainte Marie. The cathodes are formed of thin sheets of copper enclosed in porous bags, and are provided with a wooden strip at the base to prevent buckling. The flow of fresh electrolyte

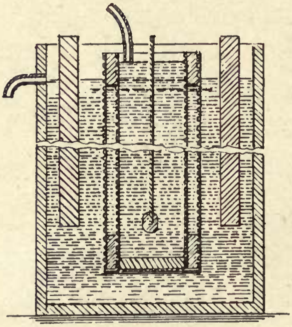


FIG. 49.—Hybinette vat for nickel deposition.

is directed into these bags by the inlet pipe, and the liquid inside the cathode compartment is maintained at a higher level than without, as shown in Fig. 49. By this means the drift of copper ions dissolved at the anode towards the cathode is checked, and only pure nickel is deposited on the copper plate within the bag, this plate being greased to facilitate stripping of the deposit.

As the deposition of the nickel proceeds, the electrolyte becomes continuously richer in copper and iron, and it flows from the cell from the run-off pipe. It is regenerated for use in the depositing cells by passing it over slabs of pure nickel or of a nickel-copper alloy. By this treatment the copper is removed by "*cementation*," and the nickel takes its place, while the iron is removed at a later stage in the process by anodic oxidation with insoluble anodes.

The Hoepfner Process at the Papenburg Works of the Allgemeine Elektro-métallurgische Gesellschaft, Germany.—Dr. Hoepfner, until his death in the year 1900, was one of the most active workers upon the problem presented by the electro-metallurgy of copper and nickel ores, and a very large number of patents were taken out in his name relating to this subject. The Papenburg Nickel Works were formerly owned by Vörster Schumann & Co., but in the years 1899—1900 the plant was transferred to a new company, and enlarged at a cost of £50,000, in order that the Hoepfner process might receive trial. This works commenced the manufacture of electrolytic nickel by the

Hoepfner process in the year 1900, and shortly afterwards they were producing the metal at the rate of $1\frac{1}{2}$ tons refined nickel per day. Although many attempts were made to introduce the Hoepfner process into other countries, these have not met with success, and so far as the author is aware at the present date, the Papenburg works is the only one where the Hoepfner process, considerably modified, is in operation. The process is cyclic, and is in principle similar to that of Browne—a mixed solution of calcium, nickel and copper chlorides being used as electrolyte, and the regeneration of the solution taking place *outside* the vats by passing the liquid from the anodic

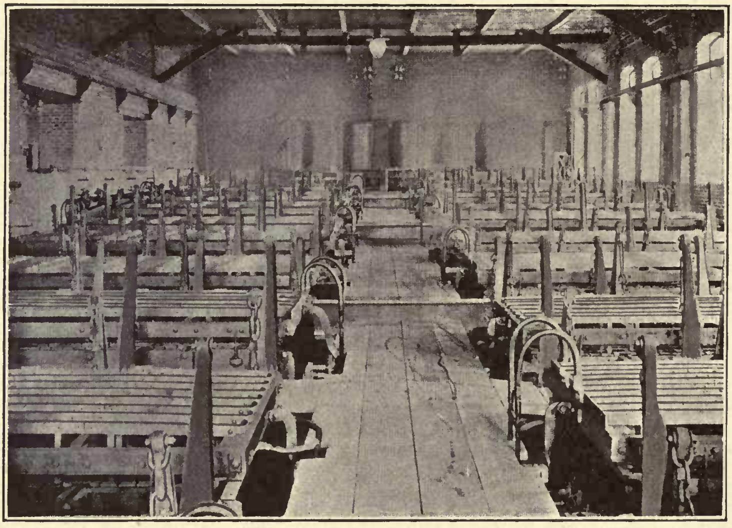


Fig. 50.—Vat room at Papenburg nickel works.

compartment of the vats over the roasted ore and matte. The copper and nickel salts are reduced to the lower state of oxidation by this leaching operation. The larger portion of the copper is now removed from this solution by electrodeposition as in the Browne process, using carbon anodes and copper cathodes. The final traces of copper are removed by chemical treatment.

The purified solution is now run into the nickel depositing vats, and this metal is recovered by electrolysis. The chlorine ions set free in the anodic compartment of both vats of depositing cells are utilised in raising the metallic salt to the higher stage of oxidation, and thus no free chlorine is evolved under normal working conditions. In this respect the Hoepfner process differs from that of Browne. The only information of recent date relating to the Papenburg works is, that the smelting plant is to be removed to Emden on the coast in order to save transport charges.

III. Dry Processes.

The Clergue-Horry Process and Furnace.—This electric furnace method of treating the nickel-iron ores of the Sudbury district was operated for some time upon an experimental basis at Sault Sainte Marie in Canada; and proposals were made for the erection of a large plant for operation of the process upon a scale of great magnitude.

The details of the treatment were as follows:—The ore was first roasted to reduce the sulphur contents under 7 per cent., and it was then pulverised and mixed with lime and carbon. The mixture was now smelted in an electric furnace of the revolving type, an alternating current being used for heating the charge. The best results obtained in the experimental trials gave an alloy containing 40 per cent. nickel, 28 per cent. iron, 12 per cent. carbon, 15 per cent. silicon, and 5 per cent. of other unestimated elements. It does not appear from the published tests that the alloy contained any copper. Krupp of Essen was reported to be interested in this electric furnace process for the treatment of the ferro-nickel ores of the Sudbury district of Ontario, and no doubt if the large works spoken of had been established, some portion of the product would have been shipped to Essen. However, so far as the writer is aware, the scheme for founding a large works using the Clergue-Horry process and furnace at Sault Sainte Marie was not carried out, and it is only recently that the proposal to establish a nickel-iron smelting works at this place has been revived in an altered and modified form.

The Heroult Furnace and Process.—The experimental trials of the Heroult electric smelting furnace at Sault Sainte Marie in Canada in the early part of 1906, of which some details have been given in Chapter X., were extended to cover the smelting of roasted pyrrhotite from the mines of the Lake Superior Corporation. This ore contained 45·8 per cent. iron and 2·23 per cent. nickel, with silica, lime, and magnesia.

In the trial run with this ore, 7,336 lbs. of ferro-nickel alloy were produced, with a power consumption of ·389 e.h.p. year per ton of alloy. The alloy produced contained from 3·70 to 4·1 per cent. nickel. As a result of this trial Dr. Haanel states that the smelting of roasted pyrrhotite in the Heroult furnace is capable of immediate commercial application. Proposals have therefore been made for the erection of a large furnace plant on the Heroult system at Sault Sainte Marie, and if the necessary capital can be raised, the establishment of a large ferro-nickel electric smelting works at this spot will be proceeded with.

CHAPTER XIV.

SODIUM.

I. Introduction.

The production of sodium by the electrolysis of fused sodium hydrate in recent years has grown into an important industry, and the older chemical method of manufacture has been quite supplanted by the newer electrolytic process. Installations of this character are now working in England, America, France and Germany. The larger number of these use the Castner cell and process, while in a few cases a modified form of this cell is employed. Reference was made in Chapter I. to the separation by electrolysis of the alkali metals, sodium and potassium, for the first time by Davy in the year 1807. It is interesting to note that the method in use to-day for the production of sodium upon a large commercial scale is similar in all respects to that used by Davy one hundred years ago. The description given by Davy of the results obtained when electrolysing caustic potash may be repeated here in his own words.

"A small piece of pure potash, which had been exposed for a few seconds to the atmosphere so as to give conducting power to the surface, was placed upon an insulated disc of platina, connected with the negative side of the battery of 250 plates in a state of intense activity, and a platina wire connecting with the positive side (of the battery) was brought into contact with the upper surface of the alkali. . . . Under these circumstances a vivid action was soon observed to take place. The potash began to fuse at both its points of electrisation. There was a violent effervescence at its upper surface; at the lower or negative surface there was no liberation of elastic fluid, but small globules having a high metallic lustre, being precisely similar in visible character to quicksilver, appeared, some of which burnt with explosion and bright flame as soon as they were formed, and others remained and were merely tarnished and finally covered with a white film, which formed on their surface."

Thorpe, in his remarks upon this epoch in Davy's life, says:—" Sometime during the first fortnight in October, 1807, he (Davy) obtained his first decisive result, and on the 19th of November he delivered what is generally regarded as the most memorable of all his Bakerian lectures. Few discoveries of like magnitude have been made and perfected in so short a time, and few memoirs have been more momentous in result, than that which Davy put together in a few hours, and in which he announced his results to the world. His cousin Edmund Davy, who at that time acted as his assistant, relates that when he saw the minute globules of the quicksilver-like metal burst through the crust of potash and take fire, his joy knew no bounds; he actually danced about the room in ecstasy, and it was some time before he was sufficiently composed to continue his experiment.[1]

We have been so accustomed to regard electro-chemistry as a growth of modern times, that it is surprising to find that Davy in the years 1806—1807, or over 100 years ago, was delivering a course of lectures before crowded audiences at the Royal Institution, London, upon "*Electro-chemistry,*" then really a new branch of science. The constitution of the alkali hydrates had long been a puzzle to the chemists of that day, and when Davy, on November 19, 1807, informed the audience gathered to hear him deliver the annual Bakerian lecture before the Royal Society that he had solved the problem by means of electricity, and that he had separated from their hydrates two metals of extraordinary properties, to which he proposed to give the names potassium and sodium, the astonishment and excitement in scientific circles was indeed great.

Davy had employed, as already stated, the method of separating these metals which is being employed to-day for the production of sodium by the ton. The long delay which occurred between the discovery of the method by Davy and its application upon an industrial scale has been due to a cause often noted in these chapters, namely, the absence of any cheap and reliable generator of electrical energy in large quantities. Until the advent of the electric light caused the development and improvement of the dynamo, electricity could only be produced by chemical batteries, and its production in this way, on a large scale, was very costly.

---

[1] "Humphrey Davy." Century Science Series. Thorpe, 1896.

Down to the year 1892 the potassium and sodium required for scientific or commercial purposes was made entirely by chemical methods of manufacture, and even the large quantities of sodium required for the Webster aluminium process, operated at Oldbury, near Birmingham, between the years 1888—1891 (see Chapter II., p. 10), were obtained by the old reduction process. The success of electro-chemical and electro-metallurgical methods in the copper and aluminium industries, however, led chemists and engineers to turn their attention to other applications of electricity, and in the year 1893 the method of preparing sodium used by Davy in 1807 was first commercially applied at the Works of the Aluminium Company at Oldbury, near Birmingham.

The development of the industry since that date has been rapid, and, as already stated, the whole of the demand for sodium is now met by the electrolytic processes. The present annual output of sodium by these processes is probably about 5,000 tons. The market for the metal is of course a limited one, and, unless new uses are found for sodium, the manufacture can never become a very large one. Much of the sodium now produced by the electrolytic works is used up by the same firms for the production of sodium peroxide and sodium cyanide, and for reduction purposes in organic colour works. Only a small proportion of the output is placed upon the market in the metallic state.

The following are details of the cells and processes that have received industrial trial, or are now in use.

II. Descriptions of Cells and Processes.

The Castner Cell and Process.—This cell and process are in use at the Weston Point and Wallsend-on-Tyne Works of the Castner Kellner Alkali Co.; at Niagara Falls, U.S.A., by the Niagara Electro-chemical Company; at the Farbe-Werke of Meister Lucius and Brunning, Hochst-am-Maine, and at several other works in France and Germany.

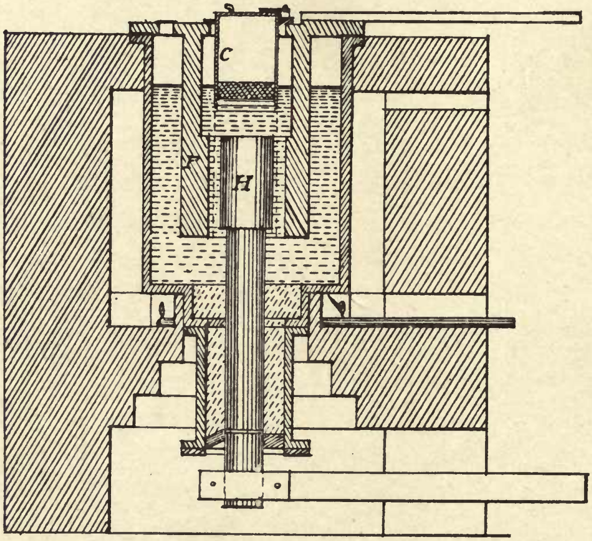


Fig. 51.—Castner sodium cell.

The original cell and process of manufacture are described in Castner's English Patent No. 13,356 of 1890, which has now expired. The cell is shown in section in Fig. 51. It was constructed of iron. The electrolyte was fused sodium hydrate free from carbonates, and was maintained at a temperature 20° C. above its fusion point, by means of external heat. The regulation of the temperature required considerable attention, since if the temperature were allowed to become too high, a portion of the metal separating at the cathode became oxidised and passed by diffusion into the electrolyte again as caustic hydrate ; while if the temperature were too low, the fused hydrate would not remain in the fluid state and ceased to act as an electrolyte. The cathode H was of iron, and projected up into the fused hydrate through the bottom of the iron vessel, into which it was cemented by cooling the electrolyte at this point. The anode F, of carbon or of metal, was connected to the under side of the cover of the vessel, and encircled the upper portion of the cathode. A small wire gauze cage C was employed to collect the metal which rose to the surface of the fused hydrate. A continuation of this cage hung as a circular curtain between anode and cathode, and functioned as a diaphragm. The metal collecting in the wire gauze cage was removed at short intervals by means of a perforated ladle, and fresh hydrate was added in order to maintain the electrolyte at a constant level. The process was practically a continuous one, and although only small cells could be employed in the manufacture, the output of metal in the early days of the industry at Oldbury reached 1 ton per day. Carrier has estimated the cost of sodium manufacture by this cell and process at $·127 per lb. (equal to 6·5*d.*), with caustic soda at $45·0 per ton, and electric power at $20·00 per e.h.p. year

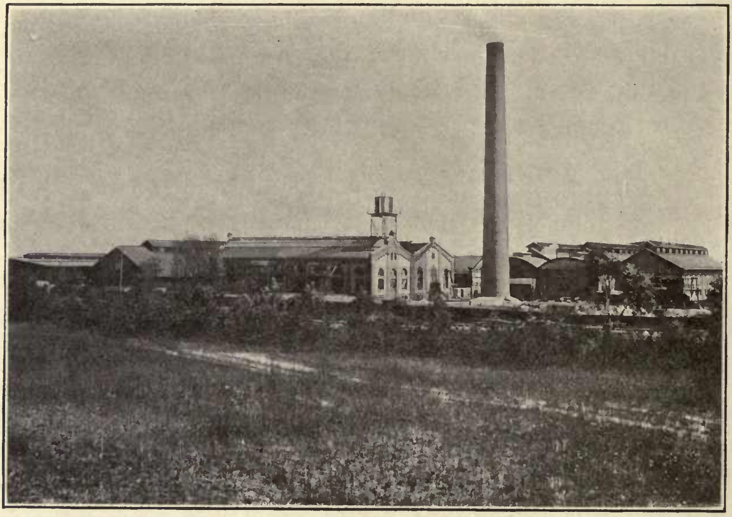


FIG. 52.—Bitterfeld Electro-chemical works (general view).

The Rathenau and Suter Cell and Process.—The Chemiche Fabrik Griesheim *Elektron* are making use at their Bitterfeld works, according to Carrier, of a modified form of the Castner cell, devised by Rathenau and Suter. The fused hydrate is contained in an iron vessel, with the anode immersed in the electrolyte as before, but the cathodes are only allowed to touch the surface of the electrolyte, and the

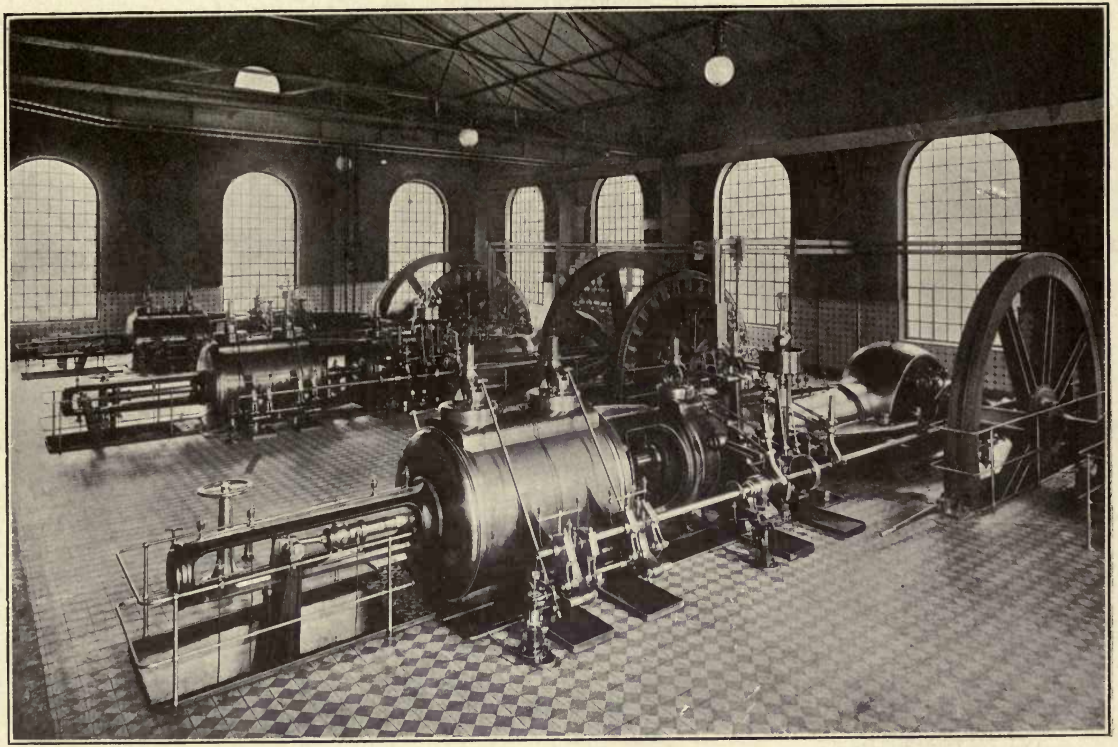


FIG. 53.—Bitterfeld Electro-chemical works (engine room).

gauze diaphragm is dispensed with altogether. The metallic sodium gathers in globules at the points where the cathodes touch the electrolyte, and is collected in a perforated ladle at very short intervals.

The Boelsterli Cell and Process.—This cell and process are reported by Carrier to be employed by the *Aluminium Industrie Aktien Gesellschaft.* The chief difference between this cell and that of Castner lies in the use of a continuous metal sheath surrounding the anode, for collection of the oxygen gas liberated at this electrode. The cathodes have rounded ends, and are arranged in a circle round the anode, with their lower ends just touching the surface of the electrolyte. The metal, as in the Rathenau and Suter cell, collects in globules around the ends of the cathodes, and must be frequently removed to prevent losses by oxidation.

The Becker Cell and Process.—Becker patented a cell and process for sodium production, in which a fused mixture of sodium hydroxide and carbonate was employed as electrolyte. The cell resembled in its main features that of Castner, but the gauze diaphragm was dispensed with, and a cone-shaped collecting cover, with its lower edges dipping on the electrolyte and negatively charged, was used to prevent oxidation of the metal floating on the surface of the fused electrolyte. This cell and process were in use at Bellegarde, in France, in the year 1900, and according to a recent review of the position of electro-chemical industry in France by Chancel, the Becker process is still employed at the works of the Société des Usines de Rivuperoux, in the valley of the Romanche.

The Darling Cell and Process.—With a view to cheapening the production of sodium, Darling in 1900 patented a cell and process, in which fused sodium nitrate was employed as electrolyte in place of sodium hydrate. By substitution of the nitrate for the hydrate, Darling hoped to obtain nitric acid from the gases liberated at the anode, and to thus double the value of the products of the electrolysis. The cell was tried upon an experimental scale at Messrs. Harrison's works in Philadelphia. The cell consisted of a cylindrical iron vessel, provided with a large cylindrical porous cup which acted as diaphragm. This diaphragm was made of two concentric and perforated steel shells, between which was packed a mixture of magnesia and Portland cement. The cathode compartment of the cell was formed by the space within the diaphragm, and contained fused sodium hydroxide. The iron containing vessel itself acted as anode, and the fused nitrate was contained in the space outsided the diaphragm cup. The gases liberated in this outer compartment of the cell were collected and conducted away for condensation as nitric acid. The metal walls of the diaphragm were protected from the nitrous vapours by making them subsidiary cathodes. According to Carrier this cell produced sodium fairly satisfactorily, but the diaphragm had a life of only 400—450 working hours, and the e.m.f. required per cell was 15 volts.

For these reasons the installation of the cell at Messrs. Harrison's works at Philadelphia was not a success, and the writer believes that no further attempt has been made to improve or modify the Darling cell and process.

The Ashcroft Cell and Process.—Ashcroft has recently patented a cell and process for sodium production, in which fused sodium chloride replaces the sodium hydrate of Castner's original process. By making use of the chloride in place of the hydrate, a much cheaper raw material is obtained, and the product obtained from the anode chamber of the cell, chlorine, becomes of marketable value.

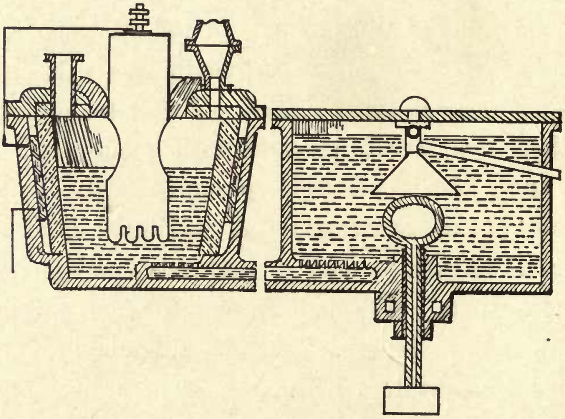


FIG. 54.—Ashcroft sodium cell.

The Ashcroft cell is necessarily more complicated in structure than that of Castner, and is formed of two sections connected by a closed channel. In the first of these the fused sodium chloride is decomposed, using a carbon anode and a molten lead cathode. The lead sodium alloy formed here is then conveyed by the connecting channel to the other section of the cell, where it is employed as anode in a molten sodium hydrate electrolyte, using a copper or nickel sphere as cathode.

The temperature required in the first section of the cell is 700° C., while in the second section a temperature of 350° C. suffices to keep the sodium hydrate in a molten state. In order to cool the lead sodium alloy as it passes from one section of the cell to the other, a device called an *equaliser* is provided between the two portions of the cell. A nickel hood is employed to collect the sodium as it rises to the surface of the second section of the cell, and by means of an overflow device at the top of the hood, the collection of the molten sodium is rendered automatic. The details of this cell and process were worked out by Ashcroft in the years 1904—1906 in conjunction with the chemists of the United Alkali Company at Widnes, England, this company having purchased the patents for the cell and process.

The industrial trial of the process is now being carried out in Norway, under the inventor's superintendence, but no details are yet available concerning the magnitude of the plant or results of these trials. Ashcroft has estimated the cost of producing sodium by his cell and process at from 2½*d.* to 4½*d.* per lb., as compared with 5¼*d.* to 7¼*d.* by the Castner process. Should this estimate prove correct, and the continuous working of the cell present no difficulties, the success of the Ashcroft process for sodium manufacture would seem to be assured.

CHAPTER XV.

TIN.

I. Introduction.

Electrolytic or electro-thermal methods have not been applied with any success to the extraction or refining of tin, for the simple reason that the ordinary metallurgical methods of extraction yield tin of a very high degree of purity. But in a subsidiary industry, namely, that of *tin stripping*, electrolytic methods have attained a position of considerable importance. The largest portion of the tin produced annually by the mines of the world is employed for the manufacture of tin-plates—that is, for the manufacture of thin sheets of mild steel coated with tin. In the working up of these tin-plates into the various vessels and utensils for which they are used, a very large proportion of waste cuttings are produced, and until electrolysis offered a simple and easy way for recovering or stripping the tin from the underlying sheet steel of these cuttings, the metal contained upon them was almost entirely lost. The thickness of the coating of tin varies with the quality of the tin-plate. The tendency in modern tin-plate mills is to use greater pressure in rolling, and to reduce the coating of tin to the thinnest dimensions that are possible with continuity of the covering. Formerly the tin-coating represented 5 per cent. of the total weight of the sheet, but this percentage has been gradually reduced until the average now is only from 1·5 to 2 per cent. The amount of tin-scrap produced is difficult to estimate, but according to a recent statement, the can factories of the United States produce from 20,000 to 30,000 tons annually, representing 400—600 tons of tin, worth, at present prices, between £70,000 and £100,000. It is probable that the similar works in Europe produce an equal amount. The price obtained for the scrap varies in different localities according to the grade of the tin sheet, and also with the price of the tin. In 1898, in New York, the average price of scrap was £5 per ton, while in 1905 it was fetching up to £11 per long ton. In Liverpool, in 1904, it was stated that clean scrap was selling at £3 per ton, while scrap from the city refuse only fetched 3*s.* per ton. The attempts made to widen the field of supply of tin-scrap by collecting the empty food cans which form a large proportion of the refuse from all large centres of population have only recently been successful, although this would appear to be a most promising field of work. In some of the larger cities of Europe and America, where a systematic collection and sorting of the city refuse is carried out, the erection of a small plant for dealing with the tin-scrap would appear a wise investment of public funds. So far, however, the writer is not aware that such a works has been started in connection with any municipal refuse plant, and only a small amount of scrap from city refuse has been worked up by the electrolytic method.

The removal of the grease, solder and enamel is necessary before these tin cans and boxes can be used for the electrolytic process. This, though adding to the cost of the raw material, does not present any special difficulties. According to Mennicke, the "*Elektron*" Works at Limehouse, in London, is the chief one using this class of raw material.

The tin stripping works now in operation are chiefly owned by private firms, and the chief difficulty in this industry is to maintain an adequate supply of the raw material. Quite recently the demand for cuttings and scrap has been increased by the adoption of a new method of removing the tin, based upon the use of dry chlorine gas. By this purely chemical method of stripping, stannic chloride is produced. The manufacture of "*tin-salts*" from tin cuttings and scrap by this method has already been taken up by some of the electrolytic alkali works, anxious to find an outlet for their chlorine.

The future of the industry is therefore somewhat uncertain, for these newer chemical methods of treatment have the merit of extreme simplicity, and there is a good market for the product—stannic chloride.

The following are details of the electrolytic tin stripping processes which have been, or are now being, operated upon an industrial scale :—

II. PROCESSES USING SODIUM HYDRATE AS THE ELECTROLYTE.

The alkaline process of tin stripping was first patented in the United Kingdom, being described in two patents granted to Beatson—No. 11,067 of 1885 and No. 12,200 of 1890. The process has been widely used, and modified by subsequent inventors.

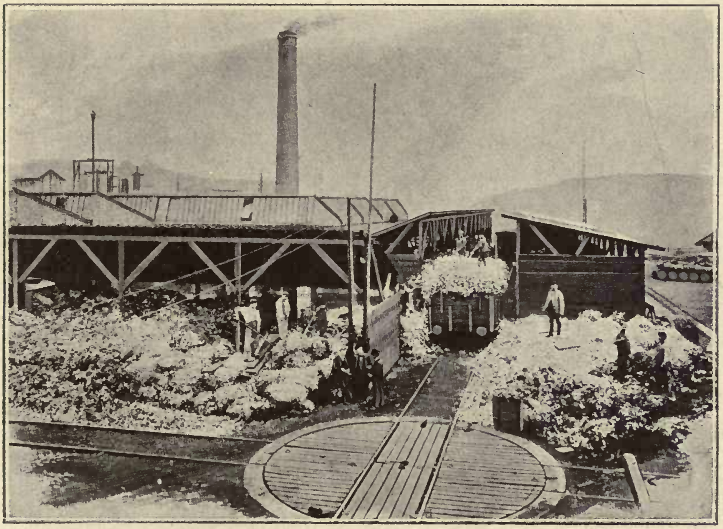


FIG. 55.—Goldschmidt tin stripping works (general view).

When tin-scrap is employed as anode material in an electrolyte of sodium hydrate, at 70° C., with nickel or tin as cathodes, the tin passes into solution as sodium stannate, and is re-deposited as metal at the cathode, while the iron remains unattacked. The process is therefore eminently suited for the recovery of the tin from tin-scrap. The solution of caustic soda need not contain more than 10 per cent. of the hydrate, and this salt is re-formed as the tin is deposited. Iron vessels may also be used to carry on the process. The chief disadvantage is, that the stripping of the tin from the scrap is not quite complete, for the iron-tin alloy which forms at the junction of the two metals is not easily dissolved. It is not found advisable in practice to push the electrolysis beyond the point where ·2 per cent. tin remains undissolved, as this would mean great waste of electrical energy. The chief works using this process is that of Th. Goldschmidt & Cie., at Essen, Germany.

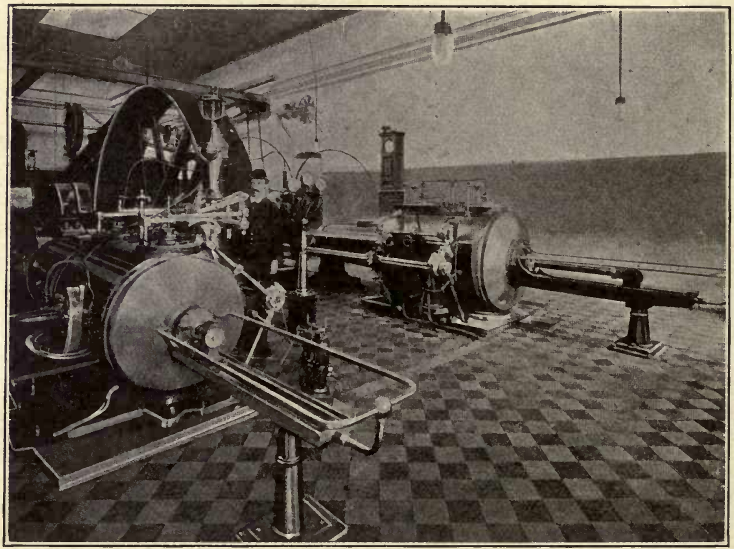


FIG. 56.—Goldschmidt tin stripping works (engine room).

This firm was one of the first to take up the recovery of tin from tin-scrap by electrolytic methods, and for a time it enjoyed a practical monopoly. The plant for tin stripping was more than once enlarged, and in 1903—1904 the capacity was stated to be 50 tons of scrap per day, equivalent to 15,000 tons per year of 300 working days. In recent years there has been a much greater competition for the scrap, and the output of the Essen works has declined in consequence.

The alkaline process of stripping has also been used, according to Mennicke, at the following places in Europe:—Uerdingen, Hanover, Nurenberg, Ludwigshafen, Emmerich, Flushing, and Pfaffstatten, and these seven factories, with that at Essen, are reported by the same authority to have treated 30,000 tons of tin-scrap annually. Similar works have also been operated at several places in the United States and in Russia. The tin stripping works now operating at Limehouse, in London, is believed to be using the alkaline method of treatment. According to Steinhart this plant is treating 10,000 tons scrap per annum.

III. Processes using an Acid Electrolyte.

A process for recovering tin from tin-scrap by aid of hydrochloric acid was patented by *F. Gelstharpe* in the United Kingdom in 1901, and was worked for a time upon a commercial scale at Manchester. The scrap was packed in wooden crates, each holding 56 lbs., and these were used as anodes, in a bath containing as electrolyte a 1·25 per cent. solution of hydrochloric acid, with a small amount of sulphuric acid. Tinned sheet was employed as cathode material, and the deposited tin, after washing, was compressed and melted down in a furnace of special design. The iron remaining after the solution of the tin was worked up into crystalline ferrous sulphate. A direct current dynamo supplying 1,500 amperes at 8 volts was employed, and four vats operated in parallel formed the stripping and depositing plant.

The plant was not a success, and only about sixty tons of tin cuttings and scrap were passed through it.

So far as the writer is aware, the use of a hydrochloric acid solution as electrolyte in tin disposition is now confined to the production of the spongy tin, known as *tin-paste*, employed in the manufacture of metallic or silvered paper. For this purpose anodes of pure tin are employed in an electrolyte of dilute hydrochloric acid of 1·025 sp. gr., and the high current density of 25 amperes per square foot is used in order to produce a spongy deposit, which floats on the surface of the vat when it is detached from the surface of the cathode. The efficiency of this process is low, on account of the hydrogen liberated with the tin of the cathode, and it is only used for the special manufacture named.

Another process for tin stripping in which an acid electrolyte was used was patented by *Smith and Gutensohn*, and was operated upon a small scale in Milan and in London in the years 1885—1890. In this case the electrolyte was composed of a 10 per cent. solution of sulphuric acid containing tin sulphate, and the scrap, packed in wooden trellis-work baskets, was used as anode, with tinned copper sheets as cathodes. The iron dissolved in this electrolyte as well as the tin, and after seven weeks' use it became fully saturated with ferrous sulphate. The trials of the process in Italy and this country were not successful, and a similar process which was operated in France also failed.

The acid processes for tin recovery are in fact difficult to operate successfully, owing to two defects. In the first place acid-resisting containing vessels are costly to construct, and are more subject to leakage than the iron vessels used for the alkali process. Secondly, the acid is not regenerated, but is converted into the corresponding salt of iron. The market for both ferrous sulphate and ferrous chloride is, however, not an expanding one. As immense quantities of these salts would be produced if the acid processes of tin stripping were used upon a large scale, there is little hope that these processes can be made to pay except where special conditions are found.

IV. PROCESSES USING VARIOUS TIN AND IRON SALTS AS ELECTROLYTE.

Claus patented a process in 1895 which depended upon the use of a 10 per cent. solution of sodium sulphide as electrolyte. The tin-scrap was employed in this solution as anode, the strength of the bath being 1·070 sp. gr., and the temperature about 90° C. A current density of 10 amperes per square foot, and an e.m.f. below ·20 volt was recommended. If the e.m.f. be allowed to rise too high, hydrogen is liberated at the cathode, in place of tin. A company was floated in the year 1896 for operating this process, and it received industrial trial in London between the years 1896—1898. The experimental trials were not successful, and renewed trials of the process at Bootle, near Liverpool, in 1900, ended likewise in failure. The latter plant was erected in order to refine impure tin containing 85 to 93 per cent. of the metal. The failure was due, not so much to any inherent defects in the process, as to the difficulty in obtaining adequate supplies of the raw material. According to Steiner about 100 tons of 90 per cent. raw tin were refined by the modified Claus process at the Bootle works, and a product of 99·9 per cent. purity was produced.

The Garcia process was patented in the United Kingdom in 1890, and was operated in London in 1891.

In this process a solution of ferric chloride was employed for dissolving the tin, the stripping operation being carried on without the aid of an electric current. The cuttings were contained in crates each holding 550 lbs., and 3 minutes' immersion in the hot ferric chloride solution was stated to be sufficient for removal of the tin. The solution of ferrous and stannous chlorides obtained in this way was filtered, carbolic and sulphuric acids were added, and the solution was then electrolysed, using anodes of carbon and cathodes of lead. The deposited tin was washed with water, and finally with petroleum, before melting down and casting into ingots. For reasons with which the writer is not acquainted, the experiments in London were unsuccessful.

The ferric chloride process has, however, been further experimented with by *Browne and Neil*, and is now being operated upon an industrial scale in America.

The scrap contained in large reed baskets each holding 200 lbs. is submitted to the action of the boiling solution of ferric chloride in large concrete vats. The moment all the tin is dissolved, the basket of cleaned scrap is withdrawn from the ferric solution, and another basket of fresh scrap is immersed. This is repeated until the whole of the ferric chloride in the stripping vat is reduced to the ferrous state. The cleaned iron is washed first with water, and then with petroleum to prevent rusting, and finds a ready sale in the iron works. The solution of ferrous and stannous chloride is run into a series of electrolytic vats, the outer cathode compartments of which are formed of concrete, the inner anode ones of porous clay. The cathode compartments are provided with cathodes of pure sheet tin, and are fed with the ferrous stannous chloride solution from the stripping vats. The anode compartments are provided with anodes of graphitised carbon, and are fed with the ferrous chloride solution, passing from the last of the cathode compartments. When electricity is passed through these vats, large crystals of pure tin are deposited at the cathode, while the chlorine liberated at the anode raises the ferrous salt to the ferric stage of oxidation. The process is, therefore, cyclic in character, and the solution flowing from the last of the anode compartments is returned to the stripping vat.

After the day's run, the tin crystals are removed from the cathode, and washed with slightly acidulated water; they are then dried, pressed into plugs, and melted. The current employed for five of these concrete depositing vats, each containing 10 porous anode compartments, is 500 amperes at 25 volts, and the yield is stated to be 34 lbs. tin per 48 to 72 e.h.p. hours. The amount of tin recovered is 40 lbs. per ton of scrap, equivalent to 1·8 per cent.

A process using *stannic chloride* solution as the solvent, and known as the *Bergsoe* process, is operated in Copenhagen, the raw material here being the tin cans, etc., gathered from the refuse of that city. The cans are used without any preliminary cleaning, a hole being cut in the bottom of each can by machinery, and the lacquered cans being separated from the others. The cans, after sorting, are placed in baskets of iron wire, and are stripped by immersing in a series of tanks through which a solution of stannic chloride is flowing. For the stripping of 10 tons of cans per day of 24 hours, 100 cubic metres of tank space are required. The stripping tanks are worked in series, and the solution flowing from the last tank is employed for the depositing vats. The solution is used cold in the stripping tanks, and in this respect the *Bergsoe* process has an advantage over the *Browne and Neil* process and alkali

MINING U. OF C. ASSOCIATION

processes. The depositing vats are worked with graphite anodes, and cathodes of pure tin sheet. The tin is deposited in crystalline form, and is easily melted down into ingot form. The chlorine liberated at the anodes raises the stannous salt to the stannic state, and thus regenerates the leaching solution. The tin is sold at the same price as Banca tin, on account of its purity. The electric current required for operating this process is stated by Schou to be only 65 e.h.p. per 1,000 kg. of tin, but without any statement of the time this figure is meaningless. A small amount of iron is dissolved with the tin at each leaching operation, and at the end of three or four months the solution becomes too saturated with ferric chloride to be longer used in the depositing vats. This feature is one of the chief disadvantages of the *Bergsoe* process. The special advantages of the *Browne and Neil* and *Bergsoe* processes, however, as compared with the alkali processes, are, that the solder and hard alloy of tin and iron are removed from the old tins by the solvent used in the stripping vats, and that the electric current required to operate these processes is only one half that required by the more generally used alkali process. The presence of this hard alloy is, as already stated, one of the chief difficulties attending the operation of the alkali processes, for, if left unremoved, it diminishes the value of the iron. For this reason, processes using chloride salts as the solvent have the advantage, and the two last-named processes may be expected in time to supplant the process now generally employed for winning tin from tin cuttings and scrap of all descriptions. Where, however, gaseous chlorine is available, and a market for tin salts exists in the locality, the purely chemical method of recovery, already described, is likely to yield the most economical results.

CHAPTER XVI.

ZINC.

I. INTRODUCTION.

SINCE the year 1883, electrical engineers and metallurgists have been experimenting with the use of electrical methods in three distinct ways in the zinc extraction and refining industry, and the literature of these attempts is now one of the most voluminous chapters of electro-metallurgical science.

The earlier attempts were directed towards obtaining aqueous solutions of the zinc salts from the roasted ores by leaching these with dilute acids; the resulting solution was then neutralised, and electrolysed with insoluble anodes, and iron or zinc cathodes for deposition of the zinc. Although a very large number of patents have been taken out in connection with this method of work, but little success has been achieved, and the writer knows of only one process of this character which is in operation upon an industrial scale at the time of writing. This is the Hoepfner zinc extraction process, which is worked as an adjunct of the ammonia-soda process at Winnington, in Cheshire, and at Hruschau, in Austria. The electro-deposition of zinc from aqueous solutions obtained, not by leaching zinc ores, but by use of zinc dust, is, however, an established branch industry, and under the name of "*electro-galvanising*" this industry has developed into one of considerable importance. The aim in this case is to obtain a protective coating of zinc upon the surface of iron or other metals, and the character of the deposited zinc is thus more important than the electrical efficiency of the process.

The second way in which electrical methods have been used in the zinc industry is for depositing zinc from fused salts, it having been found that the use of a fused salt as the electrolyte offers many advantages as compared with the use of aqueous solutions, and that the treatment of the zinc ores in order to obtain these is not so difficult as at one time was imagined. The Swinburne-Ashcroft chlorination process is the most notable of the zinc extraction processes of this type, and this is now being worked upon an industrial scale in this country.

The third way in which electricity has been applied in the zinc industry is for the direct reduction of the ores by electro-thermal methods. Although this application of electric energy for zinc extraction and refining has not yet achieved any great success, there are many who believe that at no distant date the electrical furnace will supplant the present form of furnace and methods of extraction. The losses of the metal by the older system of working are so large, varying from 10 to 30 per cent. of the metal contents of the ore, that *any* method of extraction which offered to reduce these losses to a more reasonable amount would have strong claims for adoption by the zinc smelters. In the following pages details will be given of the more important processes of extraction which have received trial in this and other countries, and in the final section of the chapter a brief account will be given of the subsidiary industry of *Electro-galvanising*.[1]

II. Wet Processes.

The general principle of all wet processes for the extraction of zinc from its ores is the same. The ore is roasted, crushed, and leached with acid or salt solutions which dissolve out the zinc, and leave the other metals behind. The liquor obtained from the leaching vats is then run through the depositing vats, when the zinc is deposited and the acid is again set free. The process is therefore a cyclic one. The difficulties attending the operation of these processes upon an industrial scale are due to the accumulation of impurities in the leaching solution, and to the fact that the deposited zinc is soluble in the regenerated liquor flowing from the depositing cells. The energy efficiency of the processes is therefore low, as a high e.m.f. is required in the end cells of the depositing system. As already stated, only one process of this character is known by the writer to be in operation, and very disastrous failures have occurred with processes which were worked upon a large scale before all the details of procedure had been adequately studied in the small scale experiments.

---

[1] The electro-magnetic and electrostatic methods of ore dressing are not dealt with in this chapter, since the subject is too extensive to be dealt with satisfactorily in a few pages.

The Hoepfner Process.—This process in 1894 was tried at Fuerfurth, in Germany, is now being worked upon an industrial scale at Hruschau, in Austria, and since 1897 at the Winnington works of Messrs. Brunner, Mond & Co. in this country.

As described in the original patent specification, the ore was first roasted and then treated with SO_2, a solution of zinc sulphite being obtained. This solution was then oxidised by blowing air through it, and the zinc sulphate was finally converted into chloride by treatment with sodium chloride after crystallisation. The electrolyte was a solution of zinc and other chlorides, lead or sodium chloride being named in this connection. The cells were divided into anode and cathode compartments by diaphragms of special construction. The cathodes were rotating discs of zinc, only partially immersed in the electrolyte. The anodes were of carbon or of lead. The rotating disc cathode was held by Hoepfner to be one of the chief features of the apparatus, since by its use treeing and other customary defects of electro-deposited zinc are avoided. Mond has patented the use of revolving rolls, each pressing against the other for deposition of zinc, and it is probable that this arrangement is used in the depositing vats at Winnington. The electrolyte is used warm. The method of preparing the zinc chloride solution is doubtful. Schnabel has stated that it is obtained by leaching the roasted ore with an alkaline solution, while according to Steinhart it is prepared by mixing zinc sulphate solution with the waste calcium chloride liquors of the ammonia-soda process. Blount states that the chloride solution is obtained by treating the roasted zinc ore with the calcium chloride solution in presence of carbon dioxide gas. This latter would yield calcium carbonate as a by-product, while the process described by Steinhart would yield calcium sulphate, which under the name of "pearl hardening" finds a ready sale to paper-makers. It is therefore probable that the latter process is the one used at the Winnington works. The chlorine set free at the anodes is employed for the manufacture of *Bleach.*

In the year 1901, 1,200 h.p. was employed at Brunner, Mond & Company's works in the Hoepfner plant, 1,663 tons of zinc and 5,000 tons of *bleach* being produced. The latest figures for the output give a daily production of 6 tons of zinc and 18 tons of bleach. The electro-deposited zinc is used chiefly in the manufacture of the brass used for cartridge cases, a very pure zinc being of advantage for this purpose.

The Ashcroft Process.—This process was patented in 1894 and 1895, and after preliminary trials had been made at Grays, in Essex, in the year 1895, it received trial upon a very extensive scale at Newcastle, New South Wales, in 1897-1898. The process was designed for treatment of the refractory lead-zinc ores of the Broken Hill mining district, New South Wales. The ores contain 30 per cent. each of lead and zinc as sulphides so intimately mixed that separation by concentrating machinery was looked upon as impossible until the oil flotation process was introduced quite recently.

In the Ashcroft process the ore was first ground and roasted, and was then leached in suitable vats, with a solution containing ferric sulphate and chloride with the corresponding salts of zinc. The iron was precipitated as ferric oxide, while the zinc passed into solution as sulphate or chloride. This ferric oxide remained partly in the leaching vats with the lead and silver, and partly separated as a precipitate on heating the effluent from the vats. The insoluble residue from the vats was smelted for recovery of the lead and silver in the usual way. The zinc salt solution, quite free from iron, was then passed through the cathode compartments of the depositing vats, and one-third of its zinc contents was obtained as metal at the cathodes.

It was then passed through the anode compartments of the same vats, iron anodes being employed in half of these and carbon anodes in the others. Solution of the anode with formation of ferrous sulphate or chloride occurred in the first of these, and the oxidation of this salt to the ferric state in the final series of vats. The effluent from the latter was then ready for use again in the leaching vats. As already stated, this process was operated first at Grays, in Essex, and then at Newcastle, in New South Wales, by a company called the *Sulphide Corporation*, with a nominal capital of £1,100,000. Although over £250,000 was expended upon the New South Wales plant and trials, these ended in failure, and, so far as the writer is aware, since 1898 the plant at this place has been devoted to other purposes. The causes of failure were stated by the inventor to be as follows:—

(1) Falling off in the metal contents of the ore sent from the extraction plant.

(2) Inability to make the process truly cyclic in character, owing to the accumulation of manganese and other impurities in the electrolyte.

(3) Insufficient financial support for the maintenance of the works upon the scale for which they were planned.

The Siemens and Halske Process.—This process was patented in Germany in 1895, and in some respects resembled the Ashcroft process. After preliminary trials at Berlin it was introduced into New South Wales by the *Smelting Company of Australia*, a company with a nominal capital of £500,000 which had purchased mining rights and erected works at Illawarra, New South Wales. A solution of zinc sulphate was obtained by leaching the crushed and roasted ore with a solution containing 5 to 10 grams per litre of free sulphuric acid. The resultant zinc sulphate solution was then deprived of its zinc by electro-deposition in vats containing lead anodes and zinc cathodes, with imperfect wooden partitions as diaphragms. When the amount of free acid in the electrolyte exceeded 10 grams per litre, the electrolyte was withdrawn from the depositing vats and returned to the leaching portion of the plant. From the fact that nothing has been heard of this process since 1898, one may safely assume that the trials at Illawarra were unsuccessful, and that the process is no longer in use.

The Cowper-Coles Process.—This process followed the same lines as that of Siemens and Halske, a solution of sulphuric acid being employed to leach the roasted ore, and the zinc sulphate solution found in this way being then electrolysed in vats containing lead anodes and rotating zinc or aluminium cathodes. The process was patented in 1896, and a small syndicate—the Cowper-Coles Zinc Extraction Syndicate—was formed to finance its experimental trial at Hayle, in Cornwall, in the years 1897—1898.

The trials at the latter place did not result in financial success, and so far as the writer is aware, the Cowper-Coles zinc extraction process is not now in operation at any works in this country, or in Australia, although the British Broken Hill Proprietary Company had some thought of introducing it into New South Wales in the year 1898.

Other wet processes which have received industrial trial are those of *Dieffenbach*, at Duisburg, in Germany, an unknown process at Hernek, near St. Petersburg, of *Gunther*, at Furfurt-on-the-Lahn, and of *Nahnsen*, at Lipine, in Silesia. None of these are believed to have resulted in a permanent installation of plant for zinc extraction by the wet process. As pointed out in the introduction to this chapter, the process of Hoepfner is the only process which has achieved permanent success. This is due chiefly to the fact that it is worked in connection with the ammonia-soda process, and assists in the utilisation of a chemical (calcium chloride) which would otherwise be a waste product.

III. Dry Processes.

The Swinburne-Ashcroft Process.—The original Swinburne process was an electrolytic one, and differed from the preceding ones in that fused zinc chloride was employed as electrolyte in place of an aqueous solution of a zinc salt, while the chlorine was used for the entirely novel purpose of decomposing the original ore at a high temperature in a special vessel called the "*Transformer.*" This method of attacking the problem of the separation of metals from refractory mixed sulphide ores was first patented by Swinburne in 1897, and has since been developed by this inventor in conjunction with A. E. Ashcroft and the *Phœnix Process Company*, of London.

A large amount of experimental work has been carried out in connection with the development of this process, first in London, and then at Milton, in Staffordshire. The process has been somewhat modified since its first inception, and it is now being worked upon a fairly large scale of operation at the Works of the Castner Kellner Alkali Company at Weston Point, Lancashire, a plant with a capacity of 30 tons ore per week having been installed at that place. Owing to the great importance of any successful method of dealing with refractory sulphide ores of zinc, lead, and silver of low metal contents, the Swinburne-Ashcroft process will be described at some length. The description of the process and apparatus is drawn chiefly from the articles and papers which have been published from time to time by the two inventors, James Swinburne and A. E. Ashcroft.

The original process, as patented in British Patent Nos. 10,829 and 10,829A, of 1897, was as follows:—

The crushed but unroasted ore was fed into an externally heated cauldron containing molten lead and molten zinc chloride. The silver of the ore alloyed with the lead, the zinc and lead sulphides mixed with the zinc chloride, and formed a molten mixture of lead and zinc sulphides and chlorides. After removal of the lead silver alloy the molten mass was electrolysed, and first the lead and then the zinc was obtained in the molten state at the cathode, while sulphur escaped as vapour at the anode. No chlorine was liberated, and the zinc chloride was finally recovered from the gangue, etc., of the original ore by distillation. Owing to the metals being separated at a temperature above their melting points, no difficulties from spongy deposits occurred. The cost of operating this process was estimated at 38*s.* per ton of ore in Australia, or at 25*s.* per ton if only the silver and lead were extracted.

The Phœnix Process (*Parent*) *Company*, with a nominal capital of £50,000, was floated in London in 1898 to finance the early trials with this process, and a small experimental plant was erected at Milton, in Staffordshire.

Here trials of the process upon a small industrial scale were carried on until 1901, when the plant at Milton was dismantled and sold, and the trials of the process were transferred to the works of the Castner Kellner Company at Weston Point, Lancashire. Numerous modifications have been introduced into the process since the early laboratory trials in London and date of the first patent specification. As now worked at Weston Point, the process is as follows:—[1]

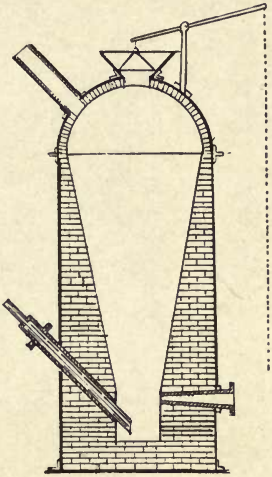


Fig. 57.—Swinburne-Ashcroft transformer for zinc extraction.

The crushed ore is run into a vessel called the transformer, which in appearance resembles a small blast furnace made of fireclay, with iron outside or of lined iron. The transformer runs continuously. The top of the transformer has a cone like a small blast furnace; the ore is poured into the fused chlorides, and at the same time chlorine is blown in. The chlorine enters at the bottom by a carbon tube, which is a sort of tuyere. The cold part of this tube is connected to an iron pipe, which brings the chlorine. The chlorine bubbling through the fused bath displaces the sulphur, which comes off and is condensed. The action of chlorine on the sulphides in question evolves a great deal of heat, so that the transformer is self-heating, and no coal or coke firing or outside heating is required. The temperature is controlled by the rate of admission of the chlorine. If the temperature be too low chloride of sulphur may be formed; this may also be formed if there is a deficiency of ore supply. On the other hand, if the chlorine and ore are

---

[1] Abstracted from Swinburne's "Faraday Society Paper," June 30th, 1903.

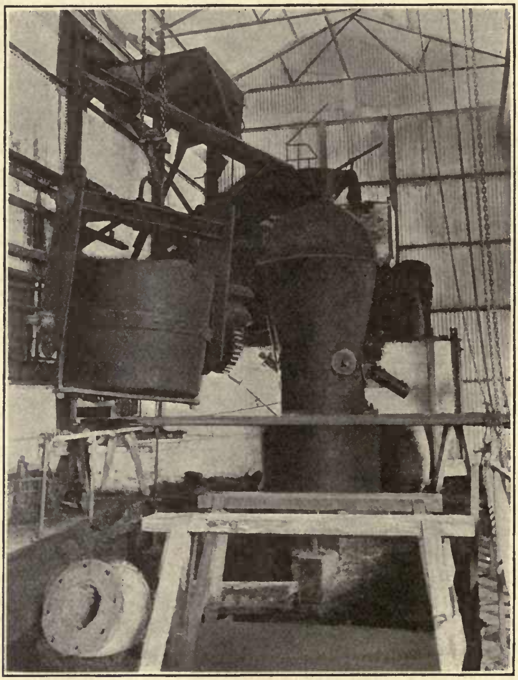


FIG. 58.—Swinburne-Ashcroft transformer for zinc extraction.

supplied too quickly, the transformer will get too hot, and some of the chlorides will distil over and be condensed with the sulphur. By running the transformer fast the temperature can be raised so high that all the chlorides distil over, leaving only the gangue. It has been found best to employ large transformers of from 10 up to 15 tons capacity. The regulation of the transformer working needs no skill. A hole is opened at the top of the transformer, and as the apparatus is connected through the sulphur chamber to a chimney, there is a gentle suction. This draws air in at the hole which burns the sulphur vapour, and the blue flame shows if there is enough ore. When too little ore is fed, fumes of ferric chloride will fill the top of the transformer and pass into the sulphur. This must be carefully avoided, and there should be excess of sulphides in the transformer until the end of the charge. At the end the ore feed is stopped, and the transformer run till brown fumes appear; these are by-passed round the sulphur chamber. The charge is then run out and allowed to cool.

At first great difficulty was experienced in pumping the chlorine, as a pressure of some 10 to 15 pounds on the inch is needed when the transformer is nearly full. The pumping now presents no difficulty, and a chlorine pump will run day and night continuously without trouble. The intermediate or chemical stage of the process varies with the ore used. The fused mass from the transformer may contain chlorides of lead, zinc, iron, manganese, copper, silver and gangue. It is run into water, and through a filter press when cool enough. This takes out the gangue and lead chloride carrying most of the silver. This gangue is easily separated from the lead and silver chlorides. These chlorides are then dried and fused in contact with lead, which extracts the silver and any gold, and then with zinc, which gives lead, practically pure, and anhydrous neutral zinc chloride, which is ready for the electrolytic vats.

The filtrate from the press contains a little lead and silver in solution, and copper, iron, manganese, and zinc. The lead and silver are taken out with spongy copper. The copper is taken out as sponge or "cement" by zinc, and we have left iron, manganese, and zinc chlorides.

The iron is chlorinated up to the ferric state, and zinc oxide is added to cause precipitation. This throws down hydrated ferric oxide, which is the base of iron paint, and is marketable, its value depending on the colour obtained. The solution is further chlorinated in presence of more zinc oxide, and the manganese goes down as peroxide. The final result of this chemical treatment is that zinc has been substituted for all the other metals and their chlorides. The solution of zinc chloride is then evaporated down carefully and fused. This treatment decomposes some of the chloride and produces an oxychloride. Steinhart has proposed to evaporate in vacuo in order to produce neutral anhydrous zinc chloride. Swinburne and Ashcroft have, however, found that with cautious boiling down there is not much oxygen in the fused salt, and this oxygen is got rid of in open vats, in which a preliminary electrolysis occurs using inexpensive anodes. The anhydrous neutral chloride from these vats is then added to that from the lead chloride substitution, and is taken to the final electrolytic vats.

These vats are internally heated, as in the case of aluminium. The cathode is fused zinc, and the anode is carbon. Some makes of carbon do not last well; but suitable carbons seem to stand permanently. The vats are kept under a very light suction, so that if there are any leaks, air goes in instead of chlorine coming out. The carbons are not hot enough to be burnt by this small admission of air. The vats are kept warm by the excess of the electrical power over the chemical, and the larger the vat the lower the electromotive force. A 3,000 ampere vat was first employed at Weston Point; this means a very small vat, for the output of fused salt vats is enormous, in comparison with aqueous work. A 10,000 ampere vat is now being tried. Above that size the current will be inconvenient. With the small vats 4 volts per cell is needed; with 10,000 amperes 3 is expected to suffice.

The vats are very simple in construction, being merely iron cases lined with fire-brick. The chloride soaks into the porous brick and solidifies, producing a vat with zinc chloride walls. The third step in the complete process, namely, the electrolysis of the fused zinc chloride and the recovery of the zinc chlorine, is not carried out at Weston Point, since the object of adopting the process there was to find an outlet for the surplus chlorine of the electrolytic alkali plant. Instead of electrolysing the zinc chloride it is therefore sold, and a good market is reported to exist for this product.

Fig. 57 is a sectional elevation of the transformer, while Fig. 58 is a photograph of the transformer erected in the zinc extraction plant at the Castner Kellner Alkali Company's works at Weston Point.

The Steinhart and Vogel Process.—This process was worked out, in order to provide a method of recovering metallic zinc from the zinc oxide, produced by the Fry and Everitt process for smelting mixed sulphide ores. The latter process was owned by a company called the *Smelting Corporation*, floated in London in 1898, with a capital of £1,100,000, and a works was erected in the following year at Ellesmere Port, on the River Mersey, for operation of this process upon a large scale. Over £250,000 was expended upon the erection and equipment of this works. The smelting process was based upon the fusion of the mixed sulphide ore with ferric oxide and sodium sulphate, and calcination of the product from this fusion in Siemens regenerative gas furnaces.

The works failed from two causes, one being the excessive emission of sulphurous acid vapour from the smelting cupolas, and the other, incorrect statements as to the completeness of the separation of the zinc from the lead and other metals of the ore. However, the following details of the electrolytic portion of the process as worked out by Steinhart and Vogel may be of interest and value.

The zinc oxide as delivered by the Smelting Company was dissolved in commercial hydrochloric acid; the resultant solution was filtered to remove the lead, iron, and other impurities, and was then concentrated partly by steam coils. The dehydration of this concentrated zinc chloride solution was carried out under reduced pressure (26 inches to 27 inches of mercury vacuum) by external gas or coke heating. The final step in the process, the electrolytic decomposition of the fused anhydrous zinc chloride, was effected in closed stoneware cells, externally heated and provided with carbon electrodes, the cathode connections being along the bottom of the cell where the molten zinc collected. The electrolyte was maintained at a temperature of 500° C., and an e.m.f. of 4 to 5 volts was required to work the cell when using a current of 600 amperes. Over 3 cwts. of zinc were produced in the experimental trials of this cell, with a current efficiency of 91½ per cent. The results of the experimental trials at Messrs. Steinhart and Vogel's laboratory in London, and later at one of the works of the United Alkali Company in Widnes, Lancashire, proved that this process could be successfully operated under the following conditions:—

(1) Vitreous stoneware for the cell and cover, with as few joints as possible.

(2) Gas-firing.

(3) Carbons shaped so as to produce the least waste from corrosion.

(4) Zinc-level to be constant.

The further experiments with this process were abandoned owing to the failure of the Fry and Everitt process, and closing down of the works at Ellesmere Port.

IV. Electro-thermal Processes.

The advantages offered by electro-thermal processes of zinc extraction are well brought out, if one studies the figures for the consumption of fuel by the ordinary method of reduction. The average consumption of coal *per ton* of zinc is found to be from 3 to 4 tons, and as already stated the loss of zinc during the distillation process is generally over 10 per cent., and with bad work may rise up to 35 per cent. or 40 per cent. Although no very reliable figures have yet been published for the work of the electric furnaces specially designed for zinc extraction, it has been asserted with some degree of authority that from 3 to 10 k.w. hours will suffice to obtain 1 kg. of zinc, the lower figure being that required for the purest ore. One ton of zinc by the newer method would therefore demand from 3,000 to 10,000 k.w. hours, and with cheap power (say the k.w. hour for $\frac{1}{10}$*d.*) the cost would be between £1 5*s.* and £4 3*s.* 4*d.* If, however, the electric furnace process yielded a better return of metal from the ore, as well as a purer zinc, even the higher of these limits would not be prohibitive. In the following pages the more noteworthy attempts to apply the electric furnace to zinc production are described.

Casoretti and Bertani's Furnace.—This is one of the earliest electric furnaces designed for zinc extraction, and it is shown in sectional elevation in Fig. 59. The roasted ore mixed with the reducing and fluxing materials is first heated in the usual way by coal or gas heating in the muffles A and B; the partially reduced and fused mass then passes between the electrodes where the final heating and distillation of the zinc occur. The zinc fumes are led by the canals to the condensation chamber while the slag and cinders pass away below. This furnace is therefore of the combination type. It has been tried by the Società Elettro-metallurgica Lombarda in Italy, and the kg. of zinc has been obtained in it, with an expenditure of 2 e.h.p. hours and ·15 kg. of reduction coal. The writer is not aware whether any larger installation of this furnace has been erected in Italy.

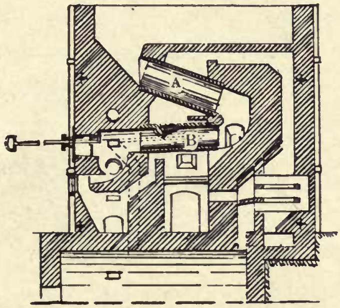


Fig. 59.—Casoretti and Bertani zinc extraction furnace.

De Laval's Furnace.—This furnace is the invention of M. Gustav de Laval, a Swedish engineer. The earlier trials of the furnace were carried out at Trollhatten, in Sweden, by a small syndicate promoted by the inventor. Over 2,000 tons of zinc are reported to have been made by the de Laval furnace and process, the product being of a high degree of purity (testing from ·03 to ·06 per cent. lead and ·01 per cent. iron), and finding a ready sale in Europe in competition with the best brands of Vielle Montagne zinc. The patents have now been purchased by a Belgian Company, the *Société*

Anonyme Mètallurgique Procedes de Laval, of Brussels, and the Trollhatten plant has been removed to Hafslund in Norway, where cheaper and larger power is available. The de Laval furnace is shown in sectional elevation in Fig. 60, and is almost self-explanatory. The finely divided zinc ore, which may be employed unroasted, is mixed with coal and the necessary fluxing materials, and is then submitted to the action of an electric arc, the ore being fed into the furnace by a plunger acting intermittently, seen on the left hand side of the furnace. The zinc, lead, gold and silver are reduced and volatilised under this treatment, and by careful regulation of the distance of the electric arc from the heap of ore on the floor of the furnace, it is stated that the purity of the product can be controlled at will. The iron combines with the sulphur and remains on the floor of the furnace. No details of the apparatus and arrangements for condensing and separating the metallic fumes have, however, yet been published, although de Laval has patented

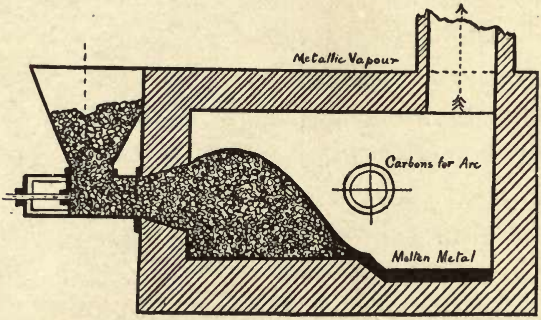


FIG. 60.—De Laval Zinc extraction furnace.

a condenser in which carbon monoxide is employed as an inert gas, to prevent oxidation of the hot zinc fumes. The e Laval furnace up to the present has been chiefly used for refining raw spelter containing up to 99 per cent. zinc, and its real value for extracting zinc from its ores has not been adequately tested. Negotiations were carried out relating to its application to the smelting of the lead-zinc sulphide ores of Saxberg, near Ludwika, in Hungary, but so far as the writer is aware, no practical developments have taken place yet, in connection with this proposal.

The Salgués Furnace and Process.—This furnace is protected by French Patent No. 330,665 of 1903, and it has been operated at the carbide works at Crampagna, in the French Pyrenees. The furnace is of the closed type, and escape of the gases is prevented by cooling the wall-joints, and openings where the electrodes enter. The zinc vapours then condense in these joints and hermetically seal them. The heating is effected by three electrodes, two placed above the third, which lies along the floor of the furnace, and is partly protected by the metal regulus that gathers around it. The ore is crushed, and roasted if desirable, before mixing with the carbon, fluxes, etc. The metal is reduced and fuses, or volatilises, according to the conditions ruling at the time. About 1 per cent. of zinc remains in the slag, and the actual losses vary from 2 per cent. up to 10 per cent., according to the richness of the ore, the average loss being 2½ per cent. with 42·5 per cent. ore. The zinc fumes are evolved with considerable amounts of carbon monoxide gas, and this circumstance renders the condensation of metallic zinc more difficult. The Salgués process and furnace at Crampagna are therefore operated to produce *zinc dust* and *zinc white*. The former is an impure form of zinc containing 95 per cent. zinc and lead, with 2 per cent. zinc as zinc oxide, while zinc white is a pigment made from zinc oxide. In order to obtain metallic zinc direct by the Salgués process and furnace, it is necessary to give the ore a preliminary roasting with carbon, whereby the greater proportion of the carbon monoxide gas is formed and passes away, before the charge enters the electric furnace. *Zinc white* is produced from hand-picked ore, and by working with a low e.m.f. the temperature is kept down, and the impurities of the raw ore are not volatilised. The conversion of the zinc vapour into zinc oxide is effected by allowing air to pass with the metallic fume from the furnace into the condensing chamber, the combination of the zinc and oxygen being marked by the production of great flames and stifling white vapour. Care must be taken during this reaction that explosions do not occur. On cooling and separating the solid portion of this vapour, it is found to be zinc white of a remarkable degree of purity and fineness, the presence of carbon monoxide with the evolved gases having contributed to this physical characteristic.

As regards the power consumption with the Salgués furnace and process, 5 kgs. zinc are stated to be obtained per k.w. day from 45 per cent. ore in furnaces of 100 k.w. capacity.

Brown and Oesterle's Process.—These two American chemical engineers have sought to combine the reduction of zinc ore and production of calcium carbide in one operation, the idea being to obtain two saleable products in place of one and thus to halve the running costs of the process. The process is based upon the smelting of a charge of zinc sulphide, lime and carbon, in definite proportion in an electric furnace, metallic-zinc, calcium carbide, carbon disulphide and carbon monoxide being produced by the chemical changes within the mass. Experimental trials with this furnace and process were made in the University of Wisconsin in the years 1903—1904, but they did not lead to any success, the trials having proved that the conditions required to produce good calcium carbide were different from those required for smelting and volatilising zinc. From the experience gained in these trials Brown and Oesterle consider that the direct electric smelting of zinc sulphide is practicable, the charge being the ore, lime and carbon in definite proportions, and the products metallic zinc, calcium sulphide and carbon monoxide. The fused calcium sulphide would be tapped in this case, and the sulphur recovered, while the whole of the zinc would pass off with the carbon monoxide gas as fume or vapour. No experimental trials of this process, however, appear to have been made by them.

W. Mc A. Johnson's Furnace and Process.—This inventor has taken advantage of the fact that the thermal efficiency of the old retort process is fairly high during the first stage of the reduction of the zinc ore, to design a furnace and process of the combination type, similar in principle to that of Casoretti and Bertani (see p. 287).

Johnson states that the first efficiency of the retort process is about four times greater than that obtained during the last stage, and working on these lines he has combined a "pre-heater" with his electric furnace.

In this combination furnace, the first stage of the reduction of the ore is effected by indirect (presumably gaseous) heating, while the last stage is carried out by direct electric heating. The mechanical difficulties connected with the design and working of a furnace of this type have now been overcome, and the inventor is only waiting a favourable time to test his furnace and process upon an industrial scale of operations. He has also designed and patented a special form of condenser for use with this furnace, which overcomes some of the special difficulties connected with the condensation of the zinc vapour to metallic zinc. The formation of zinc dust (blue powder) in place of metallic zinc is one of the chief difficulties attending the operation of the electric furnace methods of zinc extraction.

Johnson summarises the advantages of the electric furnace methods of zinc extraction as follows:—

(1) A purer and more uniform product.
(2) A higher recovery of zinc from the ore.
(3) A wider range of ores available for use.
(4) Lower labour charges.
(5) Higher thermal efficiency.
(6) Decreased capital outlay.

(7) Recovery of lead, copper, silver and gold.

Johnson's paper upon the "*Electro-metallurgy of zinc*"[1] is an interesting contribution to the literature of the subject, and in the main his opinions support those expressed by the present writer in this chapter.

V. Electro-Galvanising.

Cowper-Coles, in this country, and Alexander and Richter in Germany, were the earliest inventors to apply electro-deposition to the coating of iron or steel sheets or articles with a protective covering of zinc, or of zinc alloy, and under the name of *electro-galvanising* this method has found extended use on both sides of the Atlantic.

The electrolytic or wet method is no doubt more expensive to carry out than the older "dry" method of coating the articles by dipping them into molten zinc, but it is claimed that the coating obtained by the electrolytic process is more durable than the other, and that much finer work can be done with it.

The processes for electro-galvanising all resemble one another in principle, and differ only in minor details of the process and the plant.

A solution of a soluble salt of zinc, generally the sulphate, is used as electrolyte, and the article to be coated with zinc is made the cathode in this bath, using anodes of carbon or of lead. One of the chief difficulties in obtaining a smooth, bright and continuous deposit of the metal upon the cathode has been caused by the well-known readiness of zinc to separate in spongy form. It is now generally recognised that this spongy deposit at the cathode is due to the presence of zinc oxide in the electrolyte, and that the difficulty can be overcome by employing a solution which contains a certain amount of free acid in solution. The excess of acid must, however, be kept within certain well-defined limits, since if too great an amount of acid be present, the e.m.f. required to produce a deposit at the cathode will correspondingly increase, and a spongy deposit may be again produced from a totally different cause. In the Cowper-Coles procedure, which is that generally adopted in this country, the electrolytic bath is made up with 1 gallon of water, 35 ozs. zinc sulphate (crystalline), and $\frac{1}{10}$ oz. sulphuric acid.

---

[1] "Transactions of the American Electro-chemical Society," Vol. XI., p. 265.

The electrolyte is kept constant in composition by passing it through a vessel outside the bath, in which it percolates through a layer of zinc dust and granulated charcoal. An e.m.f. of from 3 to 5 volts between the terminals of the electrodes, and a current density of about 15 amperes per square foot of cathode surface area are employed. Messrs. Mawdslay, Sons and Field, of London, Messrs. Watson, Laidlaw & Co., of Glasgow, Messrs. Tate & Sons, of London, Messrs. Laird Bros., of Birkenhead, and Messrs. Ross and Walpole, of Dublin, have used the Cowper-Coles process at one time or another in their own works for electro-galvanising various parts of machinery, sugarboiling apparatus, ships hulls, etc., while installations of this system have also been erected in Germany and Finland. The writer is unable to say whether these plants are all still operating. Fig. 61 shows the Cowper-Coles galvanising plant at a works in Finland.

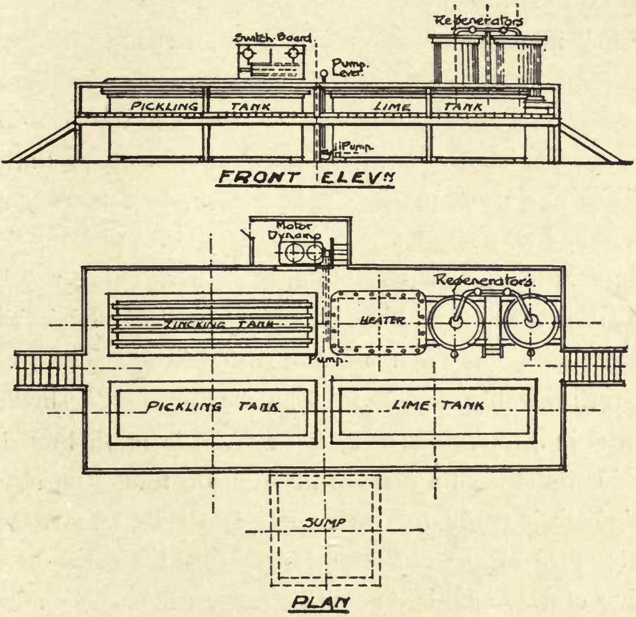


FIG. 61.—Cowper-Coles electro-galvanising plant.

The *Alexander* process has been used by the firm of Seelhorst and Werner, at Rothenfelde, in Germany. It differs from the previous process in the nature of the electrolyte. This is not merely a solution of zinc sulphate, but contains other salts in addition, such as aluminium or magnesium sulphate, *Richter's* process differs little from that of Cowper-Coles, and has been worked at Wittkowitz, in Bohemia.

In America the Electro-Galvanising Company of New York is the leading firm in this branch of electro-chemical industry.

In all of these processes not only does the temperature and chemical composition of the bath require careful and constant attention in order to get uniform results at the cathode, but especial care must be given to the cleaning of the articles before immersion in the bath. A preliminary boiling in a solution of caustic potash, to remove grease, followed by rinsing, and then a lengthy "pickling" in a dilute sulphuric acid solution, is in every case requisite. The use of the sand-blast is also sometimes advisable to effect the removal of scale and rust. After the article as been thoroughly cleaned from grease and rust, it should be placed in a tank filled with water freed from air by boiling or in water containing a little acid, until the electrolytic bath is ready to receive it. It should on no account be touched by any of the workmen with their hands after it has arrived at this stage of its preparation. The adaptation of the form of the anode to that of the object is the last point that requires attention, since the uniformity of the coat of deposited zinc will depend very largely upon the maintenance of an equal distance between all points of the anode and cathode. As regards cost, the electrolytic method of galvanising is undoubtedly more costly for large objects than the older dry method, but for small articles the difference in cost is more than proportionately reduced, and the command obtained over the thickness of the coating of zinc is of considerable importance. When the requisite conditions for obtaining a smooth, uniform and bright deposit of zinc upon the object have been complied with, the adhesion between the zinc and iron or other metal is all that could be desired.

The industry may therefore be regarded as one of the most successful of the minor electro-metallurgical industries.

GLOSSARY OF TECHNICAL TERMS AND ABBREVIATIONS.

Alternating Current.—A current changing periodically in direction.

Ampère.—The practical unit of current strength or intensity.

Anode.—The positive terminal of an electrolytic cell.

Cathode.—The negative terminal of an electrolytic cell.

Continuous Current (direct current).—A current of unvarying direction, used for electro-deposition of metals.

Current Density.—The current strength or intensity per sq. foot or sq. decimetre of electrode surface.

Diaphragm.—The porous partition or wall separating the anode and cathode compartments of the cell.

Electrode.—The terminal of the metallic or solid conductor immersed in the electrolyte.

Electrolyte.—The solution or fluid mass, through which the current passes between anode and cathode. The electrolyte must be capable of electrolytic decomposition, and, therefore, a chemical compound.

E.H.P. Hr.—Abbreviation for electrical horse-power hour.

E.H.P. Day.—Abbreviation for electrical horse-power day.

E.H.P. Yr.—Abbreviation for electrical horse-power year.

(These three are the practical units of electrical power or work. 1 E.H.P. equals 746 watts.)

E.M.F.—Abbreviation for electro-motive force, or the difference of potential which causes an electric current to flow in any conductor between two points.

H.P.—Abbreviation for horse-power, the practical unit of mechanical activity.

Induction.—The inter reaction of electro-magnetic lines of force, with the consequent production of an electric current or currents.

Ion.—An electrically charged atom or group of atoms present in any electrolyte.

Kg.—Abbreviation for kilogram, the metric unit of weight, equal to 1,000 grams. or 2·204 lbs.

K.w.—Abbreviation for kilowatt, the metric unit of electrical activity, equal to 1,000 watts, or 1·34 E.H.P.

Self-Induction.—A phenomenon observed with the flow of electric currents, analogous to the inertia of ordinary masses.

Volt.—The practical unit of electro-motive force.

Watt.—the practical unit of electrical activity, equal to the rate of work represented by a current of one ampère urged by a potential difference or E.M.F. of one volt.

NAME INDEX.

A.

Acheson, 90, 168, 228
Alexander, 293
Ashcroft, 254, 273

B.

Bastick, 76
Beatson, 259
Becker, 252
Becker & Voelcker, 157, 158
Bergsoe, 266
Berthelot, 165
Betts, 45, 58, 209, 211, 214
Betts & Valentine, 214
Berzelius, 4
Boelsterli, 252
Bradley, 31, 69
Brown, 157, 159, 161
Brown & Oesterle, 291
Browne, 237
Browne & Neil, 265
Bullier, 66

C.

Carlisle, 3
Carlson, 108
Carmichael, 135
Caro, 87
Carrier, 250, 253
Casoretti & Bertani, 286
Castner, 248
Chancel, 252
Claus, 263
Clergue & Horry, 241
Colby, 178
Cowles, 10, 14, 230
Cowper-Coles, 43, 132, 276, 293
Cruickshank, 3

D.

Darling, 253
Davy, 3, 65, 244
Davy (Robert), 65
Day & Shepherd, 164
De Laval, 287
Dieffenbach, 276
Dietzel, 57
Dumoulin, 129

E.

Easterbrooks, 56
Elkington, 6, 101, 110
Elmore, 124

F.

Faraday, 4
Fitzgerald, 91
Foerster, 234
Frank, 87

G.

Garcia, 264
Gelstharpe, 261
Gin, 177, 199, 203

Girod, 149, 201, 203
Goldschmidt, 36, 140, 179
Gore, 5
Gramme, 6
Greenawalt, 64
Gunther, 276

H.

Haag, 223
Haand, 243
Haber, 128
Hall, 7, 11, 15
Hayden, 106
Hempel, 225
Heroult, 7, 11, 17, 152, 176, 177, 183, 203, 232, 242
Hoepfner, 135, 239, 272
Hofman, 122
Hutton, 141
Hybinette, 238

J.

Jacobi, 5
Johnson, 291

K.

Keller, 144, 177, 190, 192, 203
Kiliani, 18
Kjellin, 177, 194, 203
Krause, 133

L.

Lewes, 75
Lewis, 84
Luckow, 215

M.

Machalske, 221, 228
Memmo, 70
Mennicke, 261
Moebius, 50
Moissan, 65, 140
Murray, 5
Mylius & Meusser, 165

N.

Nahnsen, 276
Nicholson, 3

O.

Oettli, 215
Ostwald, 227

P.

Parker, 76

R.

Rathenau, 217
Rathenau & Suter, 250
Richards & Roepper, 215
Richter, 293
Ronco, 234
Ross, 85

S.

Salgués, 289
Salom, 210
Schoop, 43
Secretan, 129
Siemens, 175
Siemens & Halske, 62, 275
Smith & Gutensohn, 262
Spencer, 5
Stassano, 177, 178, 203
Steiner, 264
Steinhart, 261
Steinhart & Vogel, 284
Suter & Redlich, 217
Swinburne & Ashcroft, 270, 277

T.

Taylor, 219
Thofern, 106
Thorpe, 245
Thum, 235
Tone, 227

U.

Ulke, 54, 58, 109, 110, 120, 235

V.

Vattier, 137
Vautin, 36

W.

Walker & Thum, 115
Watt, 6, 47, 101
Webster & Castner, 10, 247
Willson, 65, 153
Wohler, 65
Wohlwill, 50, 59
Woolrich, 5

Z.

Zenghelis, 39

SUBJECT-MATTER INDEX.

(See Table of Contents, pp. vii—xii.)

VAN NOSTRAND'S
"Westminster" Series

Bound in uniform style. Fully Illustrated. Price $2·00 net each.

The Volumes in the "*Westminster*" *Series* have been designed to meet the growing demand for books on practical subjects ; to bring within the ken of the non-technical reader an accurate knowledge of manufacturing processes and the practical application of modern science to industries. Each volume is written by an expert to the end that practical readers and all who are engaged in the numerous allied branches of the engineering and technical trades may have reliable works of reference. The series provides for a class not hitherto reached in published works. The volumes can be easily read by the general public, and make excellent handbooks at a moderate price for the student.

The series is well suited to public libraries and will be found valuable for libraries in engineering shops and factories.

D. VAN NOSTRAND COMPANY
Publishers and Booksellers
23, Murray and 27, Warren Streets, New York.

THE "WESTMINSTER" SERIES

Coal. By James Tonge, M.I.M.E., F.G.S., etc. (Lecturer on Mining at Victoria University, Manchester). With 46 Illustrations, many of them showing the Fossils found in the Coal Measures.

List of Contents: History. Occurrence. Mode of Formation of Coal Seams. Fossils of the Coal Measures. Botany of the Coal-Measure Plants. Coalfields of the British Isles. Foreign Coalfields. The Classification of Coals. The Valuation of Coal. Foreign Coals and their Values. Uses of Coal. The Production of Heat from Coal. Waste of Coal. The Preparation of Coal for the Market. Coaling Stations of the World. Index.

This book on a momentous subject is provided for the general reader who wishes accurate knowledge of Coal, its origin, position and extent, and its economical utilization and application.

Iron and Steel. By J. H. Stansbie, B.Sc. (Lond.), F.I.C. With 86 Illustrations.

List of Contents: Introductory. Iron Ores. Combustible and other materials used in Iron and Steel Manufacture. Primitive Methods of Iron and Steel Production. Pig Iron and its Manufacture. The Refining of Pig Iron in Small Charges. Crucible and Weld Steel. The Bessemer Process. The Open Hearth Process. Mechanical Treatment of Iron and Steel. Physical and Mechanical Properties of Iron and Steel. Iron and Steel under the Microscope. Heat Treatment of Iron and Steel. Electric Smelting. Special Steels. Index.

The aim of this book is to give a comprehensive view of the modern aspects of iron and steel, together with a sufficient account of its history to enable the reader to follow its march of progress. The methods of producing varieties of the metal suitable to the requirements of the engineer, foundryman and mechanician are described so that the worker may learn the history of the material he is handling.

Natural Sources of Power. By Robert S. Ball, B.Sc., A.M.Inst.C.E. With 104 Diagrams and Illustrations.

Contents: Preface. Units with Metric Equivalents and Abbreviations. Length and Distance. Surface and Area. Volumes. Weights or Measures. Pressures. Linear Velocities, Angular Velocities. Acceleration. Energy. Power. Introductory Water Power and Methods of Measuring. Application of Water Power to the Propulsion of Machinery. The Hydraulic Turbine. Various Types of Turbine. Construction of Water Power Plants. Water Power Installations. The Regulation of Turbines. Wind Pressure, Velocity, and Methods of Measuring. The Application of Wind Power to Industry. The Modern Windmill. Constructional Details. Power of Modern Windmills. Appendices A, B, C. Index.

Two departments of Engineering and their applications to industry form the subject of this volume: the "natural" sources of water and wind power which supply mechanical energy without any intermediate stage of transformation. Most people will be surprised at the extent to which these natural power producers are used. The widespread application of water power is generally known, but it is interesting to learn that the demand for windmills was never so great as it is to-day, and there are signs of abnormal expansion in the direction of their useful application in the great agricultural countries of the world. Though primarily of importance to the engineer, this work will be of great interest to every manufacturer who in economizing his means of power production can take the natural forces that lie to his hand and harness them in his service. The author is the son of Sir Robert Ball, the eminent mathematician and astronomer.

Liquid and Gaseous Fuels, and the Part they play in Modern Power Production. By Professor VIVIAN B. LEWES, F.I.C., F.C.S., Prof. of Chemistry, Royal Naval College, Greenwich. With 54 Illustrations.

LIST OF CONTENTS: Lavoisier's Discovery of the Nature of Combustion, etc. The Cycle of Animal and Vegetable Life. Method of determining Calorific Value. The Discovery of Petroleum in America. Oil Lamps, etc. The History of Coal Gas. Calorific Value of Coal Gas and its Constituents. The History of Water Gas. Incomplete Combustion. Comparison of the Thermal Values of our Fuels, etc. Appendix. Bibliography. Index.

The subject of this book has, during the last decade, assumed such importance that it is hoped this account of the history and development of the use of various forms of combustible liquids and gases for the generation of energy may do some service in its advancement.

Electric Power and Traction. By F. H. DAVIES, A.M.I.E.E. With 66 Illustrations.

LIST OF CONTENTS: Introduction. The Generation and Distribution of Power. The Electric Motor. The Application of Electric Power. Electric Power in Collieries. Electric Power in Engineering Workshops. Electric Power in Textile Factories. Electric Power in the Printing Trade. Electric Power at Sea. Electric Power on Canals. Electric Traction. The Overhead System and Track Work. The Conduit System. The Surface Contact System. Car Building and Equipment. Electric Railways. Glossary. Index.

The majority of the allied trades that cluster round the business of electrical engineering are connected in some way or other with its power and traction branches. To members of such trades and callings, to whom some knowledge of applied electrical engineering is desirable if not strictly essential, the book is particularly intended to appeal. It deals almost entirely with practical matters, and enters to some extent into those commercial considerations which in the long run must overrule all others.

Town Gas and its Uses for the Production of Light, Heat, and Motive Power. By W. H. Y. WEBBER, C.E. With 71 Illustrations.

LIST OF CONTENTS: The Nature and Properties of Town Gas. The History and Manufacture of Town Gas. The Bye-Products of Coal Gas Manufacture. Gas Lights and Lighting. Practical Gas Lighting. The Cost of Gas Lighting. Heating and Warming by Gas. Cooking by Gas. The Healthfulness and Safety of Gas in all its uses. Town Gas for Power Generation, including Private Electricity Supply. The Legal Relations of Gas Suppliers, Consumers, and the Public. Index.

The "country," as opposed to the "town," has been defined as "the parts beyond the gas lamps." This book provides accurate knowledge regarding the manufacture and supply of town gas and its uses for domestic and industrial purposes. Few people realize the extent to which this great industry can be utilized. The author has produced a volume which will instruct and interest the generally well informed but not technically instructed reader.

Electro-Metallurgy. By J. B. C. KERSHAW, F.I.C. With 61 Illustrations.

CONTENTS: Introduction and Historical Survey. Aluminium. Production. Details of Processes and Works. Costs. Utilization. Future of the Metal. Bullion and Gold. Silver Refining Process. Gold Refining Processes. Gold Extraction Processes. Calcium Carbide and Acetylene Gas. The Carbide Furnace and Process. Production. Utilization. Carborundum. Details of Manufacture. Properties and Uses. Copper. Copper Refining. Descriptions of Refineries. Costs. Properties and Utilization. The Elmore and similar Processes. Electrolytic Extraction Processes. Electro-Metallurgical Concentration Processes. Ferro-alloys. Descriptions of Works. Utilization. Glass and Quartz Glass. Graphite. Details of Process. Utilization. Iron and Steel. Descriptions of Furnaces and Processes. Yields and Costs. Comparative Costs. Lead. The Salom Process. The Betts Refining Process. The Betts Reduction Process. White Lead Processes. Miscellaneous Products. Calcium. Carbon Bisulphide. Carbon Tetra-Chloride. Diamantine. Magnesium. Phosphorus. Silicon and its Compounds. Nickel. Wet Processes. Dry Processes. Sodium. Descriptions of Cells and Processes. Tin. Alkaline Processes for Tin Stripping. Acid Processes for Tin Stripping. Salt Processes for Tin Stripping. Zinc. Wet Processes. Dry Processes. Electro-Thermal Processes. Electro-Galvanizing. Glossary. Name Index.

The subject of this volume, the branch of metallurgy which deals with the extraction and refining of metals by aid of electricity, is becoming of great importance. The author gives a brief and clear account of the industrial developments of electro-metallurgy, in language that can be understood by those whose acquaintance with either chemical or electrical science may be but slight. It is a thoroughly practical work descriptive of apparatus and processes, and commends itself to all practical men engaged in metallurgical operations, as well as to business men, financiers, and investors.

Radio-Telegraphy. By C. C. F. MONCKTON, M.I.E.E. With 173 Diagrams and Illustrations.

CONTENTS: Preface. Electric Phenomena. Electric Vibrations. Electro-Magnetic Waves. Modified Hertz Waves used in Radio-Telegraphy. Apparatus used for Charging the Oscillator. The Electric Oscillator: Methods of Arrangement, Practical Details. The Receiver: Methods of Arrangement, The Detecting Apparatus, and other details. Measurements in Radio-Telegraphy. The Experimental Station at Elmers End: Lodge-Muirhead System. Radio-Telegraph Station at Nauen: Telefunken System. Station at Lyngby: Poulsen System. The Lodge-Muirhead System, the Marconi System, Telefunken System, and Poulsen System. Portable Stations. Radio-Telephony. Appendices: The Morse Alphabet. Electrical Units used in this Book. International Control of Radio-Telegraphy. Index.

The startling discovery twelve years ago of what is popularly known as Wireless Telegraphy has received many no less startling additions since then. The official name now given to this branch of electrical practice is Radio-Telegraphy. The subject has now reached a thoroughly practicable stage, and this book presents it in clear, concise form. The various services for which Radio-Telegraphy is or may be used are indicated by the author. Every stage of the subject is illustrated by diagrams or photographs of apparatus, so that, while an elementary knowledge of electricity is presupposed, the bearings of the subject can be grasped by every reader. No subject is fraught with so many possibilities of development for the future relationships of the peoples of the world.

India-Rubber and its Manufacture, with Chapters on Gutta-Percha and Balata. By H. L. TERRY, F.I.C., Assoc.Inst.M.M. With Illustrations.

LIST OF CONTENTS: Preface. Introduction: Historical and General. Raw Rubber. Botanical Origin. Tapping the Trees. Coagulation. Principal Raw Rubbers of Commerce. Pseudo-Rubbers. Congo Rubber. General Considerations. Chemical and Physical Properties. Vulcanization. India-rubber Plantations. India-rubber Substitutes. Reclaimed Rubber. Washing and Drying of Raw Rubber. Compounding of Rubber. Rubber Solvents and their Recovery. Rubber Solution. Fine Cut Sheet and Articles made therefrom. Elastic Thread. Mechanical Rubber Goods. Sundry Rubber Articles. India-rubber Proofed Textures. Tyres. India-rubber Boots and Shoes. Rubber for Insulated Wires. Vulcanite Contracts for India-rubber Goods.

The Testing of Rubber Goods. Gutta-Percha. Balata. Bibliography. Index.

Tells all about a material which has grown immensely in commercial importance in recent years. It has been expressly written for the general reader and for the technologist in other branches of industry.

Glass Manufacture. By WALTER ROSENHAIN, Superintendent of the Department of Metallurgy in the National Physical Laboratory, late Scientific Adviser in the Glass Works of Messrs. Chance Bros. and Co. With Illustrations.

CONTENTS: Preface. Definitions. Physical and Chemical Qualities. Mechanical, Thermal, and Electrical Properties. Transparency and Colour. Raw materials of manufacture. Crucibles and Furnaces for Fusion. Process of Fusion. Processes used in Working of Glass. Bottle. Blown and Pressed. Rolled or Plate. Sheet and Crown. Coloured. Optical Glass: Nature and Properties, Manufacture. Miscellaneous Products. Appendix. Bibliography of Glass Manufacture. Index.

This volume is for users of glass, and makes no claim to be an adequate guide or help to those engaged in glass manufacture itself. For this reason the account of manufacturing processes has been kept as non-technical as possible. In describing each process the object in view has been to give an insight into the rationale of each step, so far as it is known or understood, from the point of view of principles and methods rather than as mere rule of thumb description of manufacturing manipulations. The processes described are, with the exception of those described as obsolete, to the author's definite knowledge, in commercial use at the present time.

Precious Stones. By W. GOODCHILD, M.B., B.Ch. With 42 Illustrations. **With a Chapter on Artificial Stones.** By ROBERT DYKES.

LIST OF CONTENTS: Introductory and Historical. Genesis of Precious Stones. Physical Properties. The Cutting and Polishing of Gems. Imitation Gems and the Artificial Production of Precious Stones. The Diamond. Fluor Spar and the Forms of Silica. Corundum, including Ruby and Sapphire. Spinel and Chrysoberyl. The Carbonates and the Felspars. The Pyroxene and Amphibole Groups. Beryl, Cordierite, Lapis Lazuli and the Garnets. Olivine, Topaz, Tourmaline and other Silicates. Phosphates, Sulphates, and Carbon Compounds.

An admirable guide to a fascinating subject.

Patents, Designs and Trade Marks: The Law and Commercial Usage.

By KENNETH R. SWAN, B.A. (Oxon.), of the Inner Temple, Barrister-at-Law.

CONTENTS: Table of Cases Cited—*Part I.—Letters Patent.* Introduction. General. Historical. I., II., III. Invention, Novelty, Subject Matter, and Utility the Essentials of Patentable Invention. IV. Specification. V. Construction of Specification. VI. Who May Apply for a Patent. VII. Application and Grant. VIII. Opposition. IX. Patent Rights. Legal Value. Commercial Value. X. Amendment. XI. Infringement of Patent. XII. Action for Infringement. XIII. Action to Restrain Threats. XIV. Negotiation of Patents by Sale and Licence. XV. Limitations on Patent Right. XVI. Revocation. XVII. Prolongation. XVIII. Miscellaneous. XIX. Foreign Patents. XX. Foreign Patent Laws: United States of America. Germany. France. Table of Cost, etc., of Foreign Patents. APPENDIX A.—1. Table of Forms and Fees. 2. Cost of Obtaining a British Patent. 3. Convention Countries. *Part II.—Copyright in Design.* Introduction. I. Registrable Designs. II. Registration. III. Marking. IV. Infringement. APPENDIX B.—1. Table of Forms and Fees. 2. Classification of Goods. *Part III.—Trade Marks.* Introduction. I. Meaning of Trade Mark. II. Qualification for Registration. III. Restrictions on Registration. IV. Registration. V. Effect of Registration. VI. Miscellaneous. APPENDIX C.—Table of Forms and Fees. INDICES. 1. Patents. 2. Designs. 3. Trade Marks.

This is the first book on the subject since the New Patents Act. Its aim is not only to present the existing law accurately and as fully as possible, but also to cast it in a form readily comprehensible to the layman unfamiliar with legal phraseology. It will be of value to those engaged in trades and industries where a knowledge of the patenting of inventions and the registration of trade marks is important. Full information is given regarding patents in foreign countries.

The Book; Its History and Development.

By CYRIL DAVENPORT, V.D., F.S.A. With 7 Plates and 126 Figures in the text.

LIST OF CONTENTS: Early Records. Rolls, Books and Book bindings. Paper. Printing. Illustrations. Miscellanea. Leathers. The Ornamentation of Leather Bookbindings without Gold. The Ornamentation of Leather Bookbindings with Gold, Bibliography. Index.

The romance of the Book and its development from the rude inscriptions on stone to the magnificent de Luxe tomes of to-day have never been so excellently discoursed upon as in this volume. The history of the Book is the history of the preservation of human thought. This work should be in the possession of evey book lover.

Van Nostrand's "Westminster" Series

LIST OF NEW AND FORTHCOMING VOLUMES.

Timber. By J. R. Baterden, A.M.I.C.E.

Steam Engines. By J. T. Rossiter, M.I.E.E., A.M.I.M.E.

Electric Lamps. By Maurice Solomon, A.C.G.I., A.M.I.E.E.

The Railway Locomotive. By Vaughan Pendred, M.I.Mech.E.

Leather. By H. Garner Bennett.

Pumps and Pumping Machinery. By James W. Rossiter, A.M.I.M.E.

Workshop Practice. By Professor G. F. Charnock, A.M.I.C.E., M.I.M.E.

Textiles and their Manufacture. By Aldred Barker, M.Sc.

Gold and Precious Metals. By Thomas K. Rose, D.Sc., of the Royal Mint.

Photography. By Alfred Watkins, Past President of the Photographic Convention.

Commercial Paints and Painting. By A. S. Jennings, Hon. Consulting Examiner, City and Guilds of London Institute.

Ornamental Window Glass Work. By A. L. Duthie.

Brewing and Distilling. By James Grant, F.C.S.

Wood Pulp and Its Applications. By C. F. Cross, E. J. Bevan and R. W. Sindall.

The Manufacture of Paper. By R. W. Sindall.

D. VAN NOSTRAND COMPANY
Publishers and Booksellers
23, MURRAY AND 27, WARREN STREETS, NEW YORK.

UNIVERSITY OF CALIFORNIA LIBRARY
BERKELEY

Return to desk from which borrowed.
This book is DUE on the last date stamped below.

MINERAL TECHNOLOGY LIBRARY

JUN 5 1953

LD 21-100m-9,'48(B399s16)476

307508

TN685
K4

Kershaw

BINDING DEPT.

UNIVERSITY OF CALIFORNIA LIBRARY